MINING METHODS AND COSTS AT THE HOLDEN MINE, CHELAN DIVISION, HOWE SOUND CO., CHELAN COUNTY, WASH.

BY JOHN R. MCWILLIAMS

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MINING METHODS AND COSTS AT THE HOLDEN MINE, CHELAN DIVISION, HOWE SOUND CO., CHELAN COUNTY, WASH.[1]

by

John R. McWilliams[2]

INTRODUCTION

This is one of a series of reports being published by the Bureau of Mines on mining methods and costs at various mines throughout the United States.

Including the Holden mine in this series is particularly appropriate at this time. The mine was operated on a low-cost, large-scale production basis to compensate for low-grade ore. Even with the best conditions and the most efficient methods, the margin of profit was small. Under the present situation of rising costs and lowering metal prices, the profit margin vanished. On June 28, 1957, the mine operations ceased; therefore, this report can review the entire life span of the mine.

ACKNOWLEDGMENTS

The author is indebted to the management and staff of the Chelan Division, Howe Sound Co., for their cooperation and assistance in preparing this report, and to the officials of the Howe Sound Co. for permission to publish the material, photographs, and data contained herein. Special acknowledgments are gratefully extended to J. S. Roper, mine manager, J. M. Newman, mine superintendent, L. R. B. Atwater, mine engineer, and R. A. Paasch, mine geologist, who gave their time and assistance freely.

LOCATION AND ACCESS

The Holden mine is on Railroad Creek near the crest of the Cascade Mountains, about 12 miles west of Lake Chelan, Chelan County, Wash. (fig. 1).

The lake, a fiordlike body of water, 1 to 2 miles in width and more than 50 miles in length, provides the only access other than by air to several small uplake settlements. Access to the mine from Chelan, the nearest supply point, is by air or regularly scheduled boat service to Lucerne, 40 miles uplake, then 12 miles west by company road to the townsite of Holden. The remote location of the mine and the severe climatic conditions pose a number of problems in logistics, maintenance, and living conditions. All supplies for the mine and campsite were brought in and ore concentrates hauled out by truck to Lucerne, thence by barge to Chelan, where they were reloaded for a 12-mile truck haul to the railhead at Chelan Falls.

An entire townsite, complete with stores, living quarters, and recreational facilities, was built near the mine to accommodate employees. In addition, a family campsite of privately owned dwellings was serviced by the company at nominal cost.

1/ Work on manuscript completed December 1957.

2/ Mining engineer, Bureau of Mines, Region I, Spokane, Wash.

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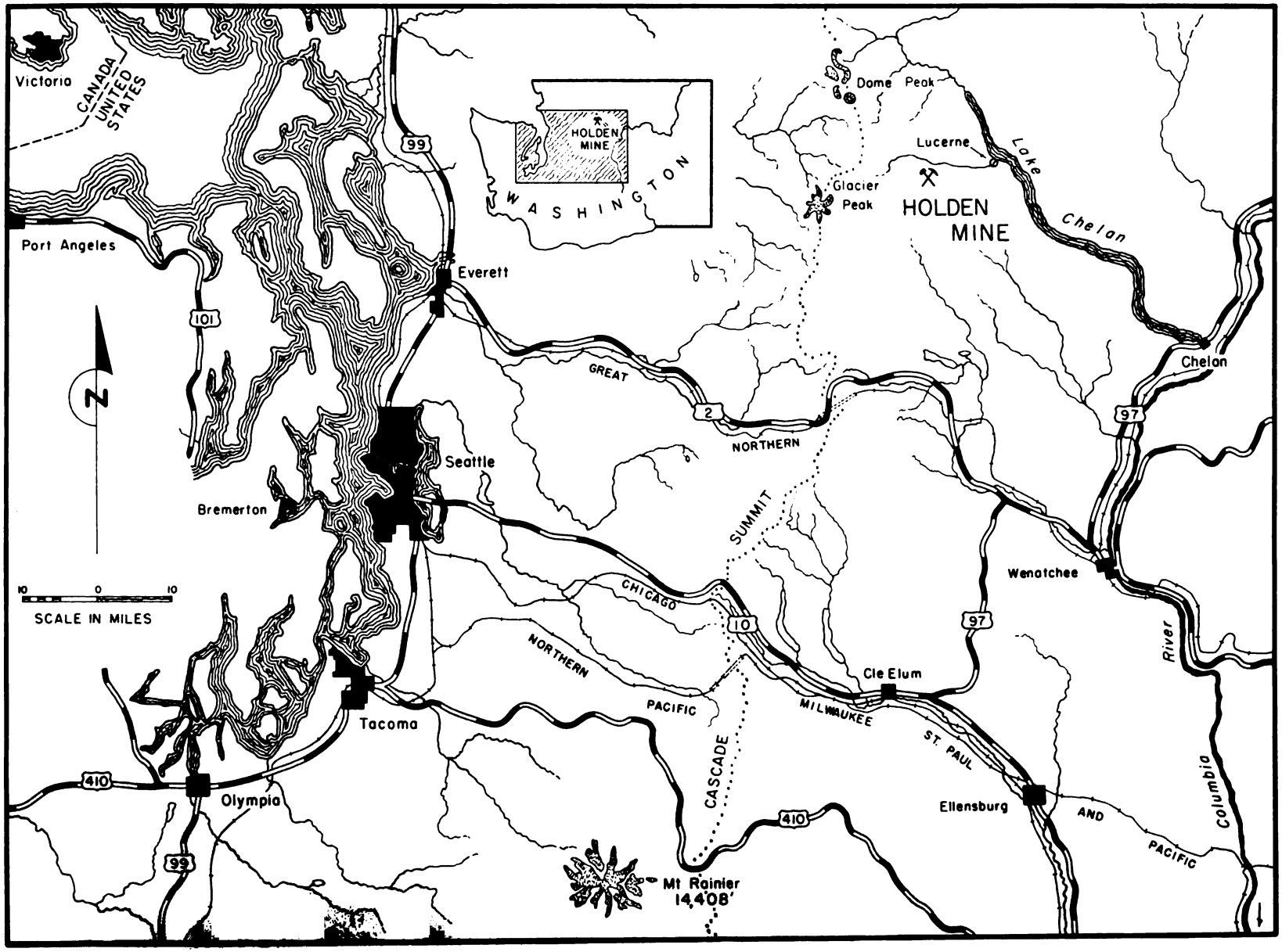

FIGURE 1. - Location of Holden Mine.

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HISTORY

The gossan outcrop of the deposit was first noticed in 1887 by a Great Northern Railway location engineer searching for a pass across the Cascade Mountains. From 1892 to 1928, many unsuccessful attempts were made to bring the deposit into production. In 1928 the property was acquired by the Britannia Mining & Smelting Co., and a subsidiary, the Chelan Copper Mining Co., was formed to explore the deposit. In 1937 the subsidiary relationship was dissolved, and the property was actively developed as an operating unit of the Howe Sound Co. During that year the concentrating plant, docks, tugs and barges, shop buildings, living accommodations (figs. 2, 3, and 4), 12 miles of road, and 54 miles of powerline were constructed. The mill was operated from April 1938 to June 28, 1957, when increasing costs of production and the lowering market price for copper finally forced closure. During this period, over 212 million pounds of copper, 40 million pounds of zinc, 2 million ounces of silver, and 600 thousand ounces of gold were produced from 10 million tons of ore.

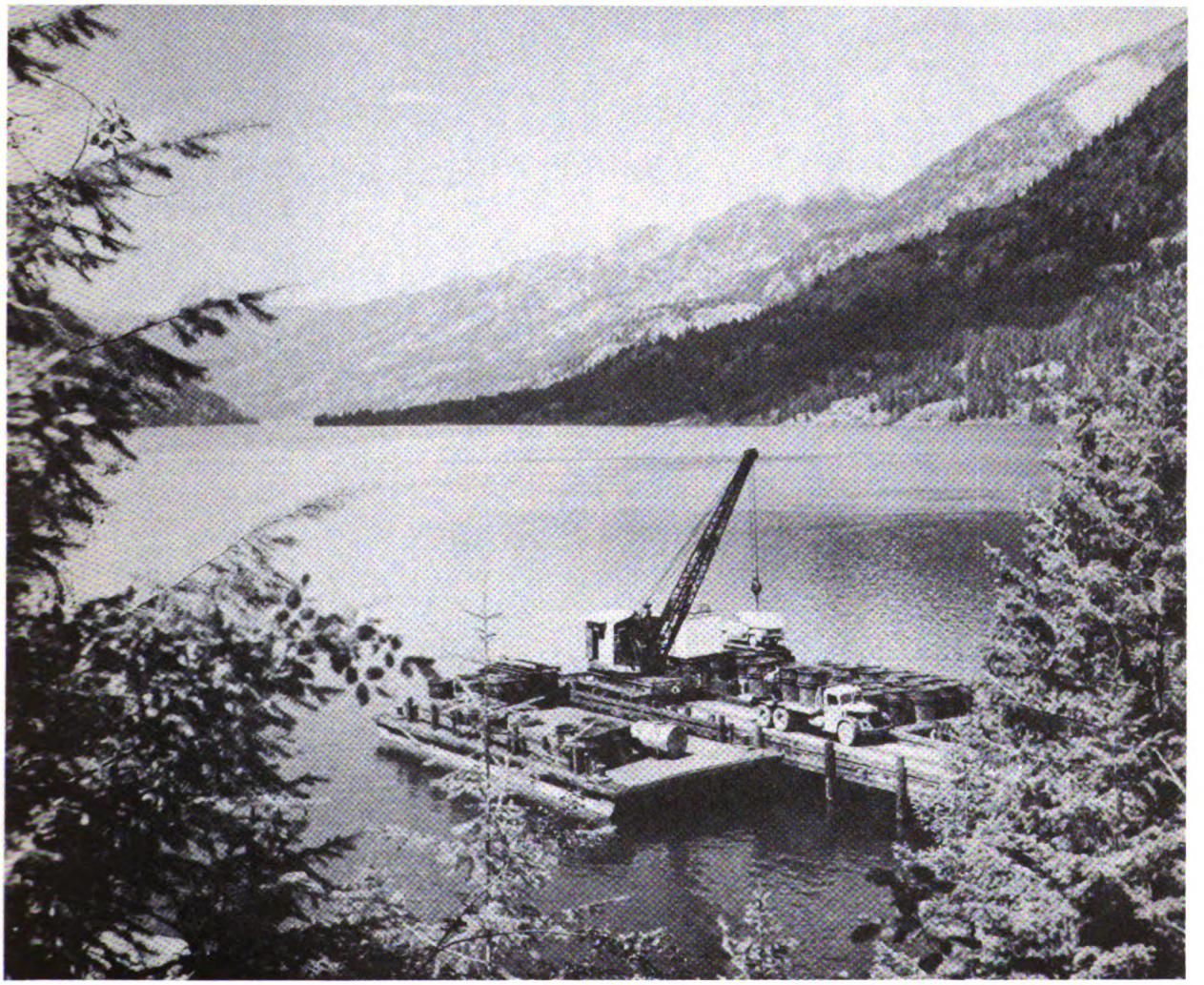

FIGURE 2. - Mine Dock on Lake Chelan. Concentrates Are Barged Downlake to Chelan.

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A

B

FIGURE 3. - A, Homes and Buildings at Holden Are Architecturally Appropriate to Surroundings: B, Large Building Houses Library, Gymnasium, Cafeteria, and Other Recreational Facilities.

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FIGURE 4. - Holden Mill and Valley of Railroad Creek.

GEOLOGY

The Holden ore body (20)3/ occurs within an extensive pyritized shear zone in metamorphosed sedimentary rocks (fig. 5). The shear zone, one of several in the area, is several hundred feet in width and of undetermined length. The ore body is about 2,500 feet in length, has economically mineralized widths up to 80 feet, and has been mined to a depth of 2,500 feet below the outcrop. On the east end of the mine commercial mineralization on the upper levels ends abruptly against a minor drag fold. The fold moderates at greater depths, and its effectiveness as an ore control is lessened. To the west the ore body splits into two fingers and gradually fades into the pyrite-pyrrhotite-biotite mineralization typical of other shear zones in the area.

The ore zone is enclosed by quartz amphibole schist of the Buckskin formation, probably of pre-Ordovician age. Mineralization was accomplished by selective replacement of siliceous phases of the host rock and was controlled by structural features, particularly the aforementioned drag fold. Within the mineralized zone hornblende has been altered to biotite, with minor chloritization and sericitization.

3/ Underlined items in parentheses refer to items in the bibliography at the end of this report.

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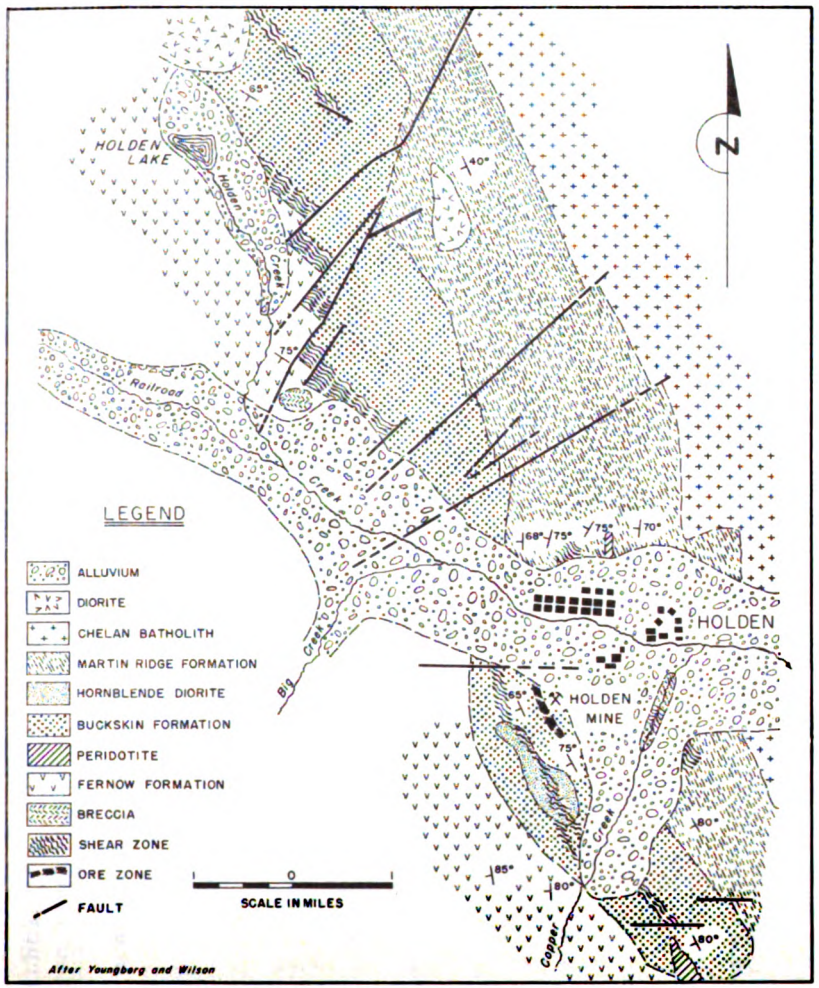

FIGURE 5. - Geology of Holden Area.

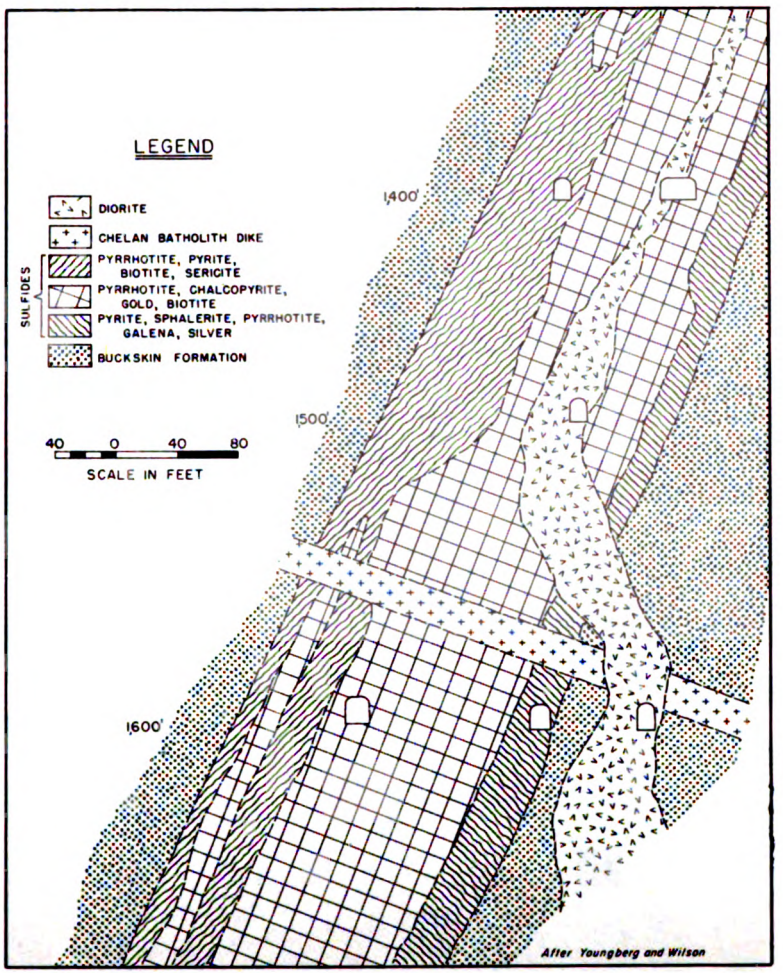

FIGURE 6. - Idealized Section of Mine Ore Zone.

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Quartz is present, as are several metallic minerals, principally pyrrhotite, pyrite, chalcopyrite, sphalerite, and gold, the latter three being the important commercial minerals. Magnetite, galena, molybdenite, and other accessory minerals are present in varying degrees of abundance. Gold is distributed throughout the ore zone roughly in proportion to the copper content. Silver is associated with sphalerite, which is concentrated along the foot wall. The foot wall is generally well-defined, but the hanging wall has assay limits (fig. 6).

Two classes of postdeposition faulting are present: Steep eastward-dipping normal faults, causing appreciable offsets to the ore body and shallow, westward-dipping thrust faults of little displacement. The one main fault in the mine is of the first type and causes an offset of 260 feet vertically and 90 feet horizontally. Minor faults of the normal type are increasingly common toward the west end of the ore body.

Several types of igneous rocks occur in the area: Peridotite, diorite, and granodiorite are the most common. Associated dikes and sills encountered in the mine are a few inches to as much as 100 feet in width. The later dikes cut almost normal to the shear; earlier diorite dikes have replaced portions of the ore zone. In general practice larger dikes within the ore zone are left as pillars, and the smaller dikes are mined with the ore.

The ore body, shear zone, and enclosing rocks have an average strike of N. 30° W. and dip 68° SW., and the ore body rakes about 60° S.

METHODS OF PROSPECTING AND EXPLORATION

Surface prospecting and exploration consisted of geological reconnaissance and mapping, supplemented by occasional diamond drilling.

Underground exploration, stope development, and general development were differentiated by purpose. The exploration work was designed to explore unknown areas and was accomplished by a combination of drifting, crosscutting, and diamond drilling. Stope development consisted of undercuts, raises, crosscuts, blasthole drifts, and other work necessary to prepare a particular stope or series of stopes for production. Other underground work that tended to benefit the entire operation was charged to mine development. This included ventilation drifts and raises, shafts, ore passes, haulageways, level development, and similar work.

Drifting and diamond drilling were integral parts of the systematic exploration of the ore body. Exploration drifts generally were driven parallel to the ore zone but up to 200 or more feet into the foot wall. Hanging-wall drifts were used originally; but, as greater depth was achieved, foot wall drifts were preferred. Drill stations were cut in the drifts at intervals of about 200 feet, and the ore zone was sampled by diamond drilling. A somewhat different approach was used to explore the downward extension of the ore body below the lowest level. Crosscuts were driven from the foot wall drift through the ore zone and 100 to 500 feet into the hanging wall. Drill stations were established in the dead end, and drillholes were fanned downward to intersect the ore zone at horizons 200 and 400 feet below the bottom level. Additional downholes were drilled to each side to give more complete coverage and a better base for evaluating the potential ore reserves.

METHODS OF SAMPLING AND ESTIMATING TONNAGES

Methods of sampling the ore body and estimating the tonnages improved as familiarity with the deposit increased. Before much was known about the physical outline

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of the ore body, drifts were run in ore as much as possible. Grab samples were taken from each drift round and checked with channel samples across the face. All openings in the ore zone, crosscuts, powder drifts, raises, etc., were sampled the same way. The ore body was sampled at 50-foot intervals by diamond-drill holes, collared in the main drifts; questionable areas were core-drilled from ore passes and powder drifts, as well. All core was logged and filed; the mineralized core was split, and one-half was filed and the other half analyzed. From the mass of accumulated data, adequate grade and tonnage estimates were calculated.

As the regularity of the footwall became known, development drifts were driven in the footwall 25 to 40 feet from the ore zone. At 50-foot intervals, 2 core-drill holes, 1 horizontal and 1 at plus 45°, were drilled through the ore body and into the hanging wall. Greater confidence in the continuity of ore values made further drilling unnecessary, except in special cases, even though the vertical distance between levels was 175 feet.

The ore outline for any block of ground was determined from a composite plan of all levels within and bordering the block and from vertical sections in the plane of the diamond-drill holes. Each section controlled a zone 50 feet wide, to which was assigned an assay value based on a weighted average of the appropriate drillhole samples. The total assay value of the entire block of ground was the weighted sum of all vertical sections, less a 10-percent dilution factor to accommodate overbreak and sloughing. Tonnage of the block was calculated by planimetering each vertical section and computing therefrom the cubic feet in each subblock. A conversion factor of 10.3 cubic feet per ton was applied to arrive at the final tonnage figure. Since the ore grade was low, considerable care had to be exercised in designing the ore outline. An uneconomically low assay-ton value could be strengthened by narrowing the ore body to exclude low-grade ore on the hanging wall. The mining method also had to be considered, as each of the several blasthole patterns had its own applications and limitations.

METHODS OF DEVELOPMENT AND MINING

General

The mining method used at the Holden mine can best be described as a modified form of sublevel stoping. Originally, breaking was accomplished by a method similar to quarry blasting with coyote holes. Undersize drifts and powder pockets were driven in the ore zone above previously prepared drawpoints and loaded with explosives. Stoping was accomplished by enormous blasts, breaking thousands of tons of ore and involving hundreds of cases of powder tightly packed into a dozen or more powder pockets and detonated simultaneously. The powder-blast method is well-suited to large-scale mining, particularly in wide stopes; but several factors, some unavoidable, contributed to its decline and eventual abandonment at the Holden mine.

The chief disadvantage of powder-blast mining was the shattering effect of the enormous blasts. The grade of ore was lowered by dilution with wall rock, and permanent workings some distance from the blast occasionally were damaged. Another factor was lack of versatility; powder-blast mining is not selective, and its efficiency decreases rapidly as the stope narrows.

During World War II experiments were started with longhole blasting - not an entirely new method but one gaining increasing acceptance with improvements in drilling equipment. At first it was used only in narrow stopes to supplement powder-blast production from larger stopes. Since 1945, all production of ore and a large part of the new development work was done by longhole blasting.

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Powder-Blast Mining

Powder-blast mining was chosen originally for two reasons: The method is economical for mining large low-grade deposits, and the Holden ore zone satisfied the basic requirement of a steeply dipping ore body of substantial width and with competent wall rock. Also, the company had already gained considerable experience in powder-blast mining at the Britannia mine in British Columbia.

A powder-blast stope was developed by a series of finger raises, culminating in a row of funnel-shaped drawholes in the floor of the proposed stope (fig. 7). Concurrently with expanding the drawholes to completely undercut the stope, sublevel drifts and powder pockets were driven in the overlying ore body at vertical intervals of about 35 feet. Stoping was begun by blasting the lowest sublevel first and continuing upward to the stope limits.

Stope Development

Preliminary work to develop a powder-blast stope involved driving 5-1/2- by 11-foot orepasses at 200-foot intervals upward from a main haulageway. From each orepass, four finger raises were driven to intersect the stope at 40-foot intervals.

Grizzly chambers, 14 by 20 feet, were cut on each finger raise about 20 feet below the stope floor and 15 feet from the foot-wall side of the ore body. Access to the chambers was through short crosscuts to a service drift in the hanging wall.

Stope undercutting was started from grizzly chambers by driving 5- by 6-foot drawhole raises at plus 40° to intersect the foot wall; drawhole wings were then extended upward along the foot wall at 45° to intersect the hanging wall. By sideswiping these raises, the entire ore block was undercut, and a flat back was left above conical funnels leading to grizzly chambers.

Coincident with undercutting, powder drifts, crosscuts, and pockets were driven in the ore body to prepare for blasting. Several powder drifts, in various stages of development, could be in progress simultaneously on as many sublevels.

Powder drifts were 4 by 6 feet in section and generally were driven in the center of the ore body at a height above the undercut not exceeding two-thirds the width of the undercut and never more than 35 feet. Powder pockets were cut along the foot and hanging walls from crosscuts to the powder drift. Powder pockets were 3 by 4 feet in section and were backfilled with muck before blasting.

Two types of powder, both ammonium nitrate base, were used in stope blasts. Pockets near the center of the stope were loaded with high-velocity powder (19,000 feet per second), and a slower powder (10,500 feet per second) was used near the walls to give the effect of a delayed shot. Both powders were packed in regular or waterproof cartridges and detonated electrically.

Powder charges were computed individually for each pocket, with due consideration to the location of the pocket, the burden, local characteristics of the ore and enclosing rock, and other factors having a bearing on the success of a blast. Maximum fragmentation was desired to minimize secondary blasting in the drawholes and grizzly chambers.

Powder-blast mining is similar to shrinkage stoping, in that all subsequent blasts fall on previously broken rock, but it differs in one important respect; all

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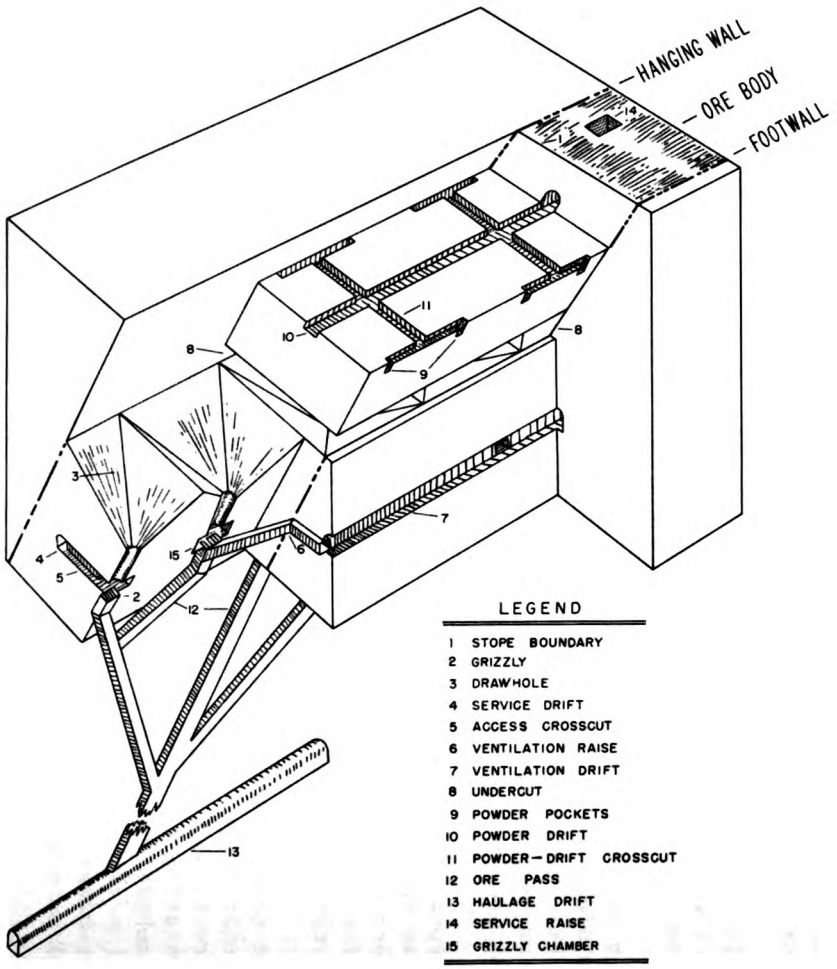

FIGURE 7. - Idealized Powder-Blast Stope Development.

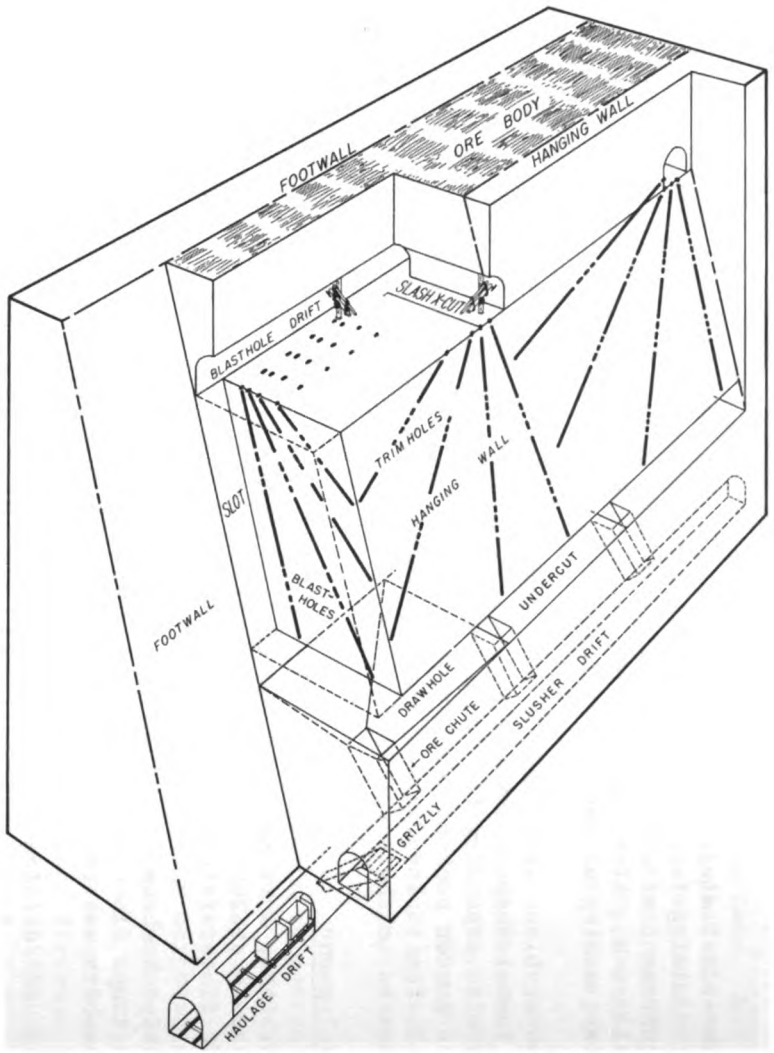

FIGURE 8. - Idealized Longhole Stope.

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preparatory work is done in solid ground above the active stope, and it is rarely necessary to enter an open stope once mining has begun. Broken rock was pulled by gravity through drawholes to the grizzly chamber, where the oversize was drilled and blasted. Undersize ore passed through the grizzly into chutes leading to the main haulage level. Secondary blasting, held to a minimum because of ventilation, consumed 0.31 pound of explosive per ton of ore mined.

The smallest powder blast used 25 cases of powder; the largest consumed 525 cases and broke 46,000 tons of ore for a powder factor of 0.57 pound per ton. The last powder blast was shot on April 22, 1945; the total 240 shots used 60,182 cases of powder and broke 3,371,701 tons of ore. The primary powder factor was 0.892 pound per ton. The overall powder factor per ton of ore drawn from stopes, to the end of 1944, including slough and overbreak, was 1.07 pounds per ton.

Longhole Mining

The longhole mining method was adapted, with some modifications, from the Noranda mine in Quebec, Canada, where it was developed and successfully used for a number of years.

Longhole mining and powder-blast mining have many points of similarity; both require the same general scheme of development for stoping and ore handling (fig. 8). Sublevel powder drifts are replaced by sublevel blasthole drifts; grizzly chambers are replaced by slusher drifts. Drawholes are essentially unchanged, finger raises are eliminated, and the use of slushers reduces the number of ore passes needed.

The resemblance continues in initial stope development, as funnel-shaped drawholes are expanded to undercut the stope. However, in longhole mining, a hanging-wall wing raise, either at the end or in the center of the stope, is continued vertically upward to form the hanging-wall slot raise. At a predetermined distance above the undercut, a foot-wall blasthole drift is driven the length of the stope, and a crosscut is extended to intersect the slot raise. The slot raise, enlarged by long downholes drilled from the crosscut, is expanded to form a vertical slot extending from wall to wall and from the blasthole drift downward to the undercut below. Longhole rings, drilled from the blasthole drift in the same plane as the vertical slot and a few feet from the free face, are blasted to remove a vertical slice of ore. Successive rings are drilled and blasted during retreat from the vertical slot to the stope limits.

As with powder-blast mining, sublevels can be developed in advance of stoping. However, longhole blasting permits several sublevels to be mined concurrently by using an inverted rill pattern. The rill method can provide faster production from a stope and a little more flexibility in ore grade, as more faces are available for mining. Broken ore may be retained in the stope as long as desired and only enough drawn to compensate for expansion, or the ore can be completely drawn except for a cushion to protect the drawholes.

Stope Development

Longhole stope development (fig. 9) was begun by driving a 6- by 8-foot orepass raise from the haulage level to the slusher drift at an incline greater than plus 45° Slusher drifts, driven the length of the proposed stope and about 20 feet below its floor, may be located in either the ore zone or the footwall, but the latter generally has shown better results. From the slusher drift, 5- by 6-foot drawhole raises on 30-foot centers were driven to intersect the footwall, where they were extended by

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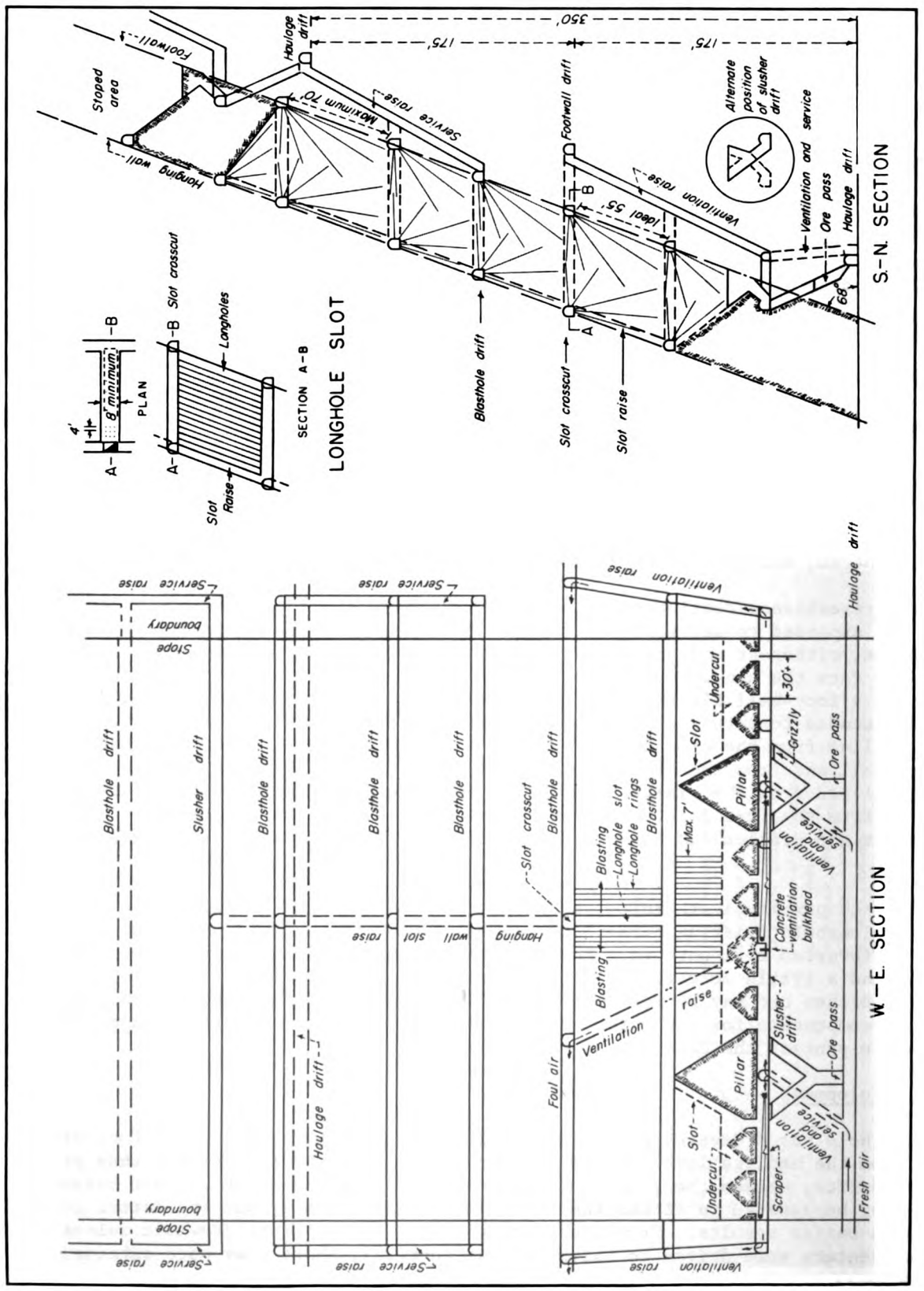

FIGURE 9. - Generalized Stope Development.

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wing raises driven along the footwall at plus 45° to 50° to intersect adjoining wings. The drawhole proper, an inverted cone, is created during undercutting. The undercut was begun at the drawhole and worked upward along the foot wall between the two wing raises and at the same time at plus 45° toward the hanging wall. The stope was undercut in this manner for its entire length and for at least 25 feet of its width. Completing greater widths became time consuming and excessively hazardous. For wider stopes, the undercut was enlarged by longhole drilling from the sublevel above.

The aforementioned slot raises were 5 by 7 feet in section and driven on the hanging wall from an extension of a wing raise. They were later slabbed out by longholes to form a vertical slot the width of the stope.

Slots may be either in the center or at one end of the stope. Center-slot stopes are usually larger and have the advantage that mining can progress simultaneously on two faces from the center to stope limits. However, in recent years, as a ground-control measure, the tendency has been toward smaller stopes; as a consequence, end slots have become more usual.

Generally, the relative amount of development work per ton of ore developed favored the larger center-slot stope. Very often, however, two end-slot stopes could be developed back-to-back with the necessary service and ventilation raises in the common pillar. In some instances it was possible to reduce development further by utilizing a single slusher drift and orepass for both stopes.

Service raises were of two types: One served a slusher drift and was driven from the nearest lower level, or, if the slusher drift and level were at the same elevation, a crosscut sufficed; the other was driven from level to level to service blasthole drifts. Both types were 6 by 8 feet in section and driven in the footwall at convenient inclinations.

The vertical interval between sublevel blasthole drifts depended somewhat on the width of the stope but was usually about 50 feet and never exceeded 70 feet. Blasthole drifts were driven along the footwall at about 0.5-percent grade from the service raise. At the end wall of the stope or in the center, as the case might be, a short crosscut was driven to intersect the previously driven slot raise. Additional crosscuts were driven from the foot-wall-blasthole drift across the ore zone and a short distance into the hanging wall at about 50-foot intervals. From these slash crosscuts, long trimholes were drilled down along the hanging wall to prevent the buildup of unbroken rock. Blasthole drifts, as the name implies, served as longhole drill stations and were expended as the stope was mined. They were driven 6 by 8 feet in section by a 2-man crew operating two 3-1/2-inch column-mounted drills. A 28- to 30-hole burn-cut pattern was standard, and a crew could muck, drill, and blast 1 round per day where no timbering was required.

MINE DEVELOPMENT

The mine was developed by more than 56 miles of underground workings over a vertical range of more than 2,300 feet from the sump, at an elevation of 2,291 feet, to the back of one of the early stopes, at 4,600 feet (fig. 10). The 1500 level, at an elevation of 3,500 feet, was the principal means of access to the mine. The hoistroom (fig. 11), collars of the underground shafts (fig. 12), and the main transportation system were at this level, as were the portal (fig. 13), mill, mine office, changehouse,and other surface structures. Fourteen working levels were used, 7 above and 6 below the 1500 level. Above the main haulageway, main levels were used principally for access and service and were spaced at vertical intervals of 100 to 250 feet.

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FIGURE 10. - Longitudinal Section in Plane of Ore Zone.

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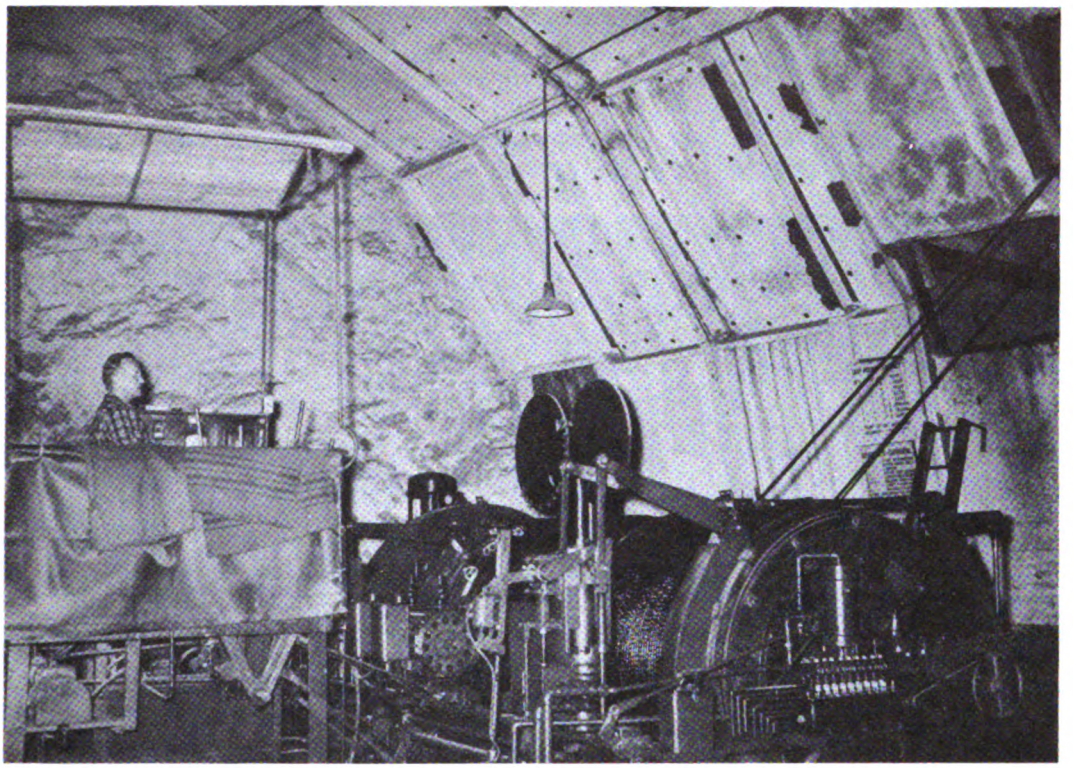

FIGURE 11. - Underground Hoist Room for No. 2 Shaft.

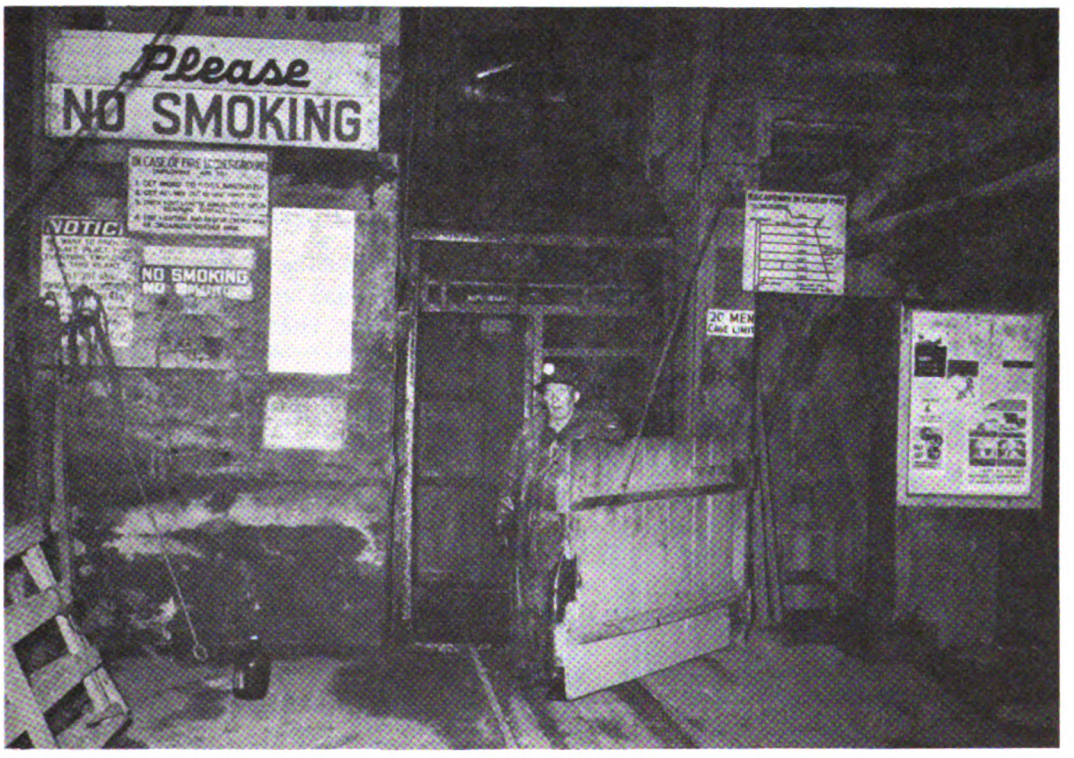

FIGURE 12. - Mancage at Main-Level of Shaft Station.

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Below the 1500, main levels were established at 175-foot intervals; every other level was a haulageway, and intermediate levels served for ventilation, service, and access. Auxiliary drifts were sometimes driven between levels to provide for greater flexibility in mining. Stopes above the 1500 level were served by the No. 1 shaft and various ore passes until this part of the mine was exhausted in 1945. Levels below the 1500 were served by the No. 2 shaft and an auxiliary winze.

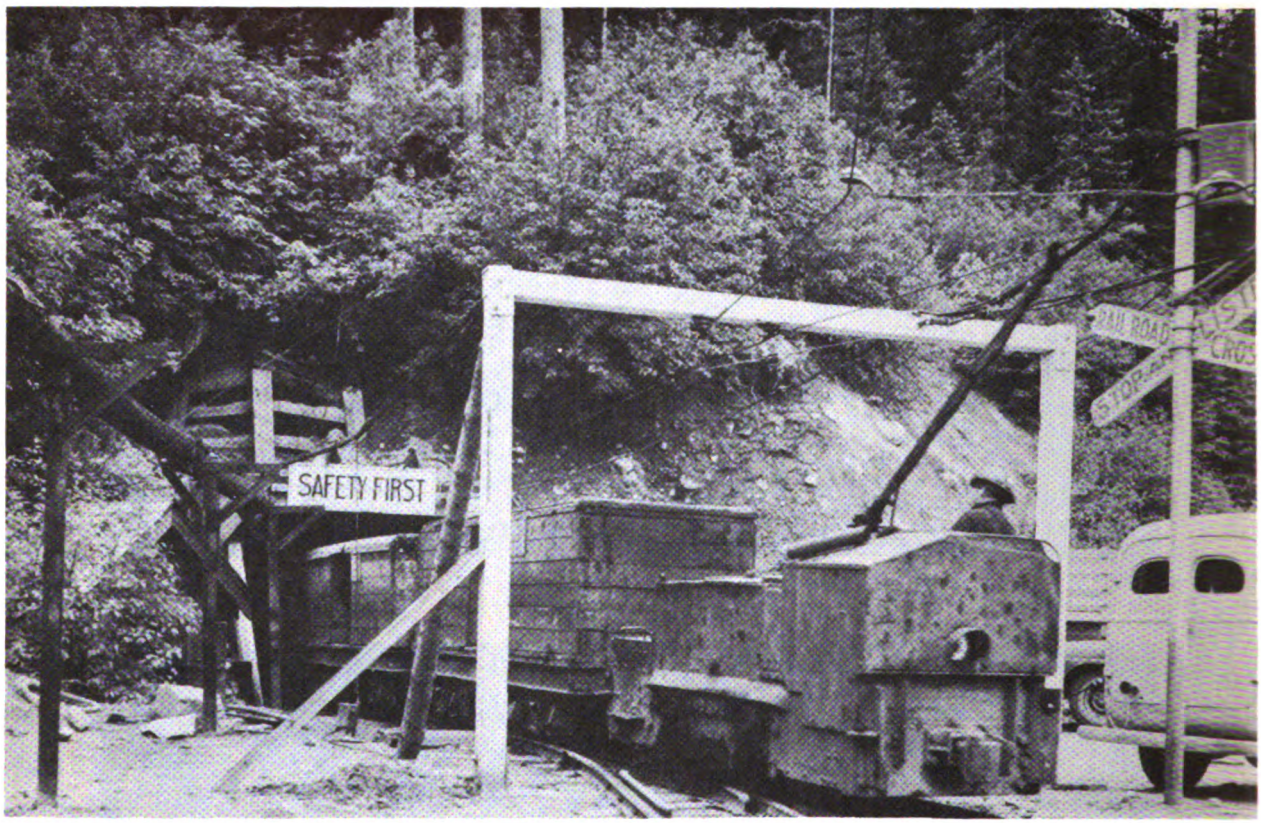

FIGURE 13. - Mantrain Entering 1500-Level Portal.

The levels were identified by elevation with respect to the outcrop, which was designated as the zero level. The lowest actual level was the 2500, although shaft stations below the 2500 were still termed levels. Other features were identified by a system based on the mine east coordinate. Stopes were arbitrarily numbered by dividing their center coordinate by 20. Raises and winzes were similarly identified by the coordinate at which they were collared. Duplication, which might occur on different levels, was avoided by varying the last digit.

Two underground crushers served the area below the main haulage level. The first was on the 1950 level and handled ore from stopes between that level and the 1500. The second crusher was installed below the 2500 level to serve the intervening area. Underground storage at the crushers in both coarse- and crushed-ore pockets could sustain the mill at full capacity for about 2 days.

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Development Details

Shafts

The No. 1 shaft was completed in 1938 and used until 1945 to service the 1250, 1100, 800, 700, and 550 levels. It was driven as a two-compartment vertical raise simultaneously from the 800 and 1500 levels and was used only for service and access. Ore from the upper levels was handled through orepasses to chutes on the 1500 level.

The No. 2 shaft, completed in January 1947, had five compartments and served all levels below the 1500. It measured 18 feet 10 inches by 12 feet 6 inches outside 10-inch timbers (fig. 14), and had two hoisting compartments, a cage compartment, manway, and a service compartment. The cage was capable of handling ore cars or a small trammer.

The No. 2 shaft was driven as a two-compartment, timbered pilot raise from the 1600 to the 1500 level, then stripped and slabbed out to full size by diamond-drilled longholes. The holes, drilled from the top down the length of the pilot raise, were loaded and shot in 8 to 24-foot rounds. Between the 1600 and 1775 levels, where the distance was too great for a single-stage drillhole, two intermediate drill stations were cut in the pilot raise. Holes were drilled, loaded, and blasted as before. From the 1775 to the 1950 level it was planned to drill, blast, and timber in three 60-foot stages. Excessive drill wander in the first stage necessitated slabbing along one rib to maintain the shaft plumb. Longholing was discontinued, and the shaft sunk to the 1950 level by percussion drilling and blasting just ahead of timbering.

Below the 1950 level the shaft was continued full size by conventional sinking methods, using a 40-hole bench round on alternate sides of the shaft. With this type of round, the drilling and loading face was always free of water and the bench easy to clean, and there was always a sizable sump with no special effort. The No. 2 shaft was bottomed at an elevation of 2,291 feet.

The No. 3 shaft was collared as an exploration heading near the east end of the mine to explore below the 2500 level. It was abandoned after extending two sets below the level.

The 283 winze, extending down from the 1500 to the 2125 level, was a three-compartment inclined shaft, originally driven for exploration purposes and later used to service the lower levels. It was driven in the footwall 75 feet from the ore body at minus 65°. The outside timber dimensions were 5 feet 4 inches by 14 feet 8 inches; timbers were 8-inch posts with 2-inch lagging on the hanging wall and between the manway and 2 skipways (fig. 15). The 46-hole V-cut round used in sinking this winze averaged 4.4 feet advance for 140 rounds. Each foot of advance required 23.33 pounds of 40-percent gelatin dynamite or 5.67 pounds per cubic yard of solid rock.

Haulage Drifts

Haulage drifts were 9 by 10 feet, except at the 1500 level, which was 10 by 10 feet, with 0.5-percent grade in favor of the load. Timbering was required only part of the time; 9-foot sets of round timber and 3- by 10-inch lagging were used where necessary. Rock bolts won increasing favor during the later years; both the wedge type, manufactured in company shops, and commercial expansion-shell bolts were used.

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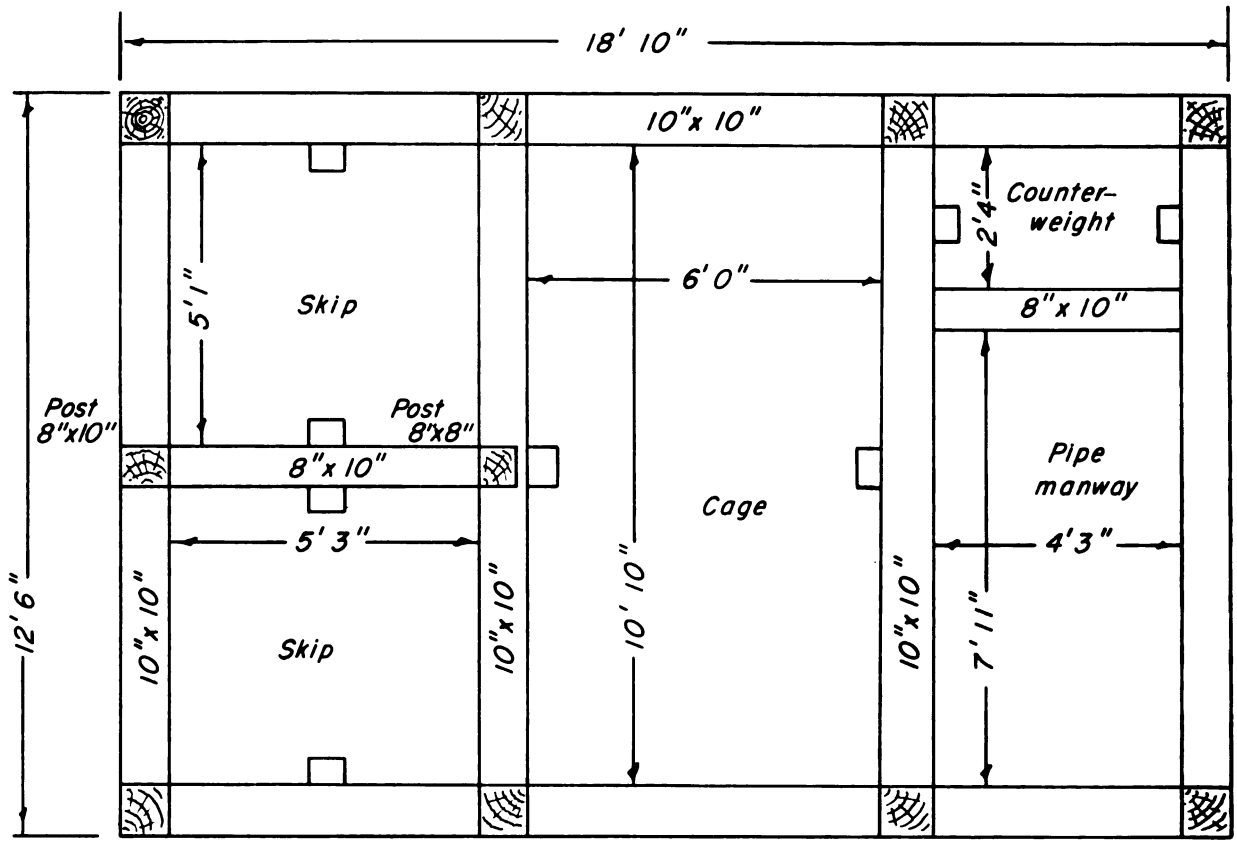

FIGURE 14. - No. 2 Shaft Timber.

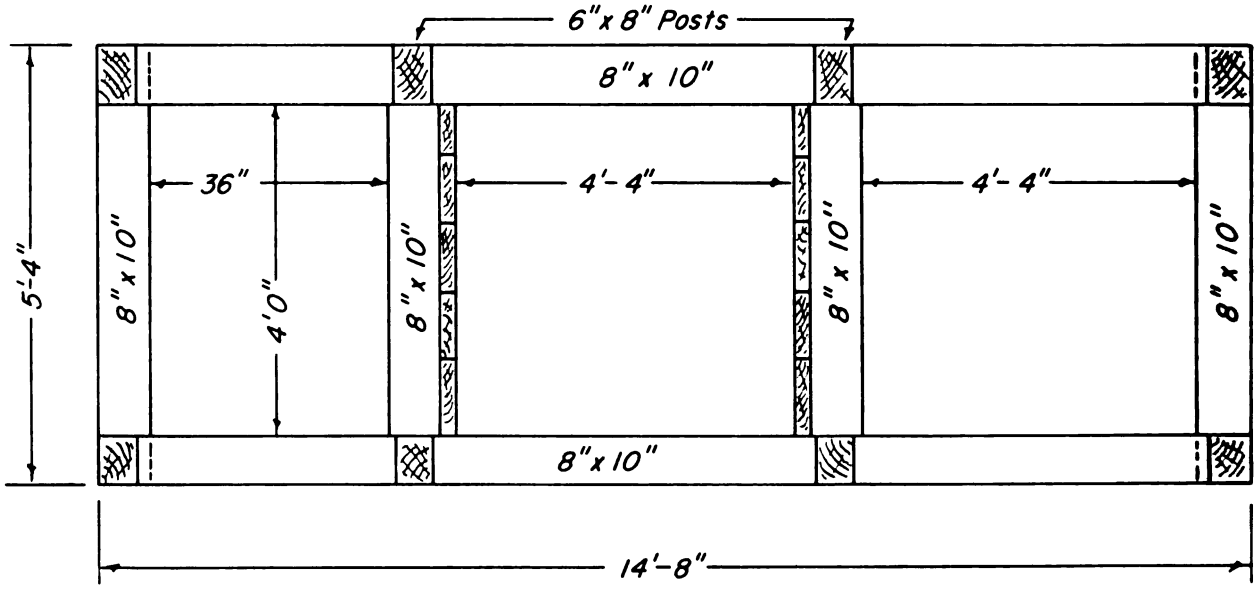

FIGURE 15. - 283 Winze Timber.

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Haulage drifts originally were driven using a 3-man crew on 3 shifts. Twin 3-1/2-inch jumbo-mounted drifters (fig. 16) were used to drill out forty 7-foot holes (fig. 17) in an 8-hour shift to average 93 feet of drilling per man. The most recent practice was to use a two-man crew on the cycle of breaking, mucking, and placing permanent pipe, ditch, and track. The crew averaged 4-1/2 rounds of 6-1/2 feet each in 6 days, or 29.25 feet advance per week. Mucking was by overshot machine into 3-ton Granby-type cars, pulled by 3-ton trolley locomotives.

Service Drifts and Crosscuts

Service drifts and crosscuts were formerly driven 5 by 7 feet in section by 1-man crews drilling and blasting one 5-foot, 19-hole round per shift. Mucking was by hand or machine into 1-ton cars. Such drifts were later driven 6 by 7 feet in section to provide room for 2 drillers and use of an overshot mucker.

Service Raises

Two main service raises above the main level supplemented the No. 1 shaft. One three-compartment inclined raise, with an ore chute, waste chute, and manway, served the 1400 and 1250 levels; the other, a two-compartment incline, also served the 1100 level. These raises speeded stope development on the upper levels by providing additional working faces. The early raises were 8 by 8 feet in section, driven 5 by 8 feet and later sideswiped to size. A 2-man raise crew could complete a 7-foot, 22-hole round in 1 shift. Later service raises were 5 by 7 feet for general use, 6 by 8 feet to service slusher drifts, and 5-1/2 by 11 feet for main ventilation raises and main ore passes.

MINING

Breaking Ore

When blasthole mining was first adopted, longholes were drilled entirely by diamond drills. A shortage of skilled drillers and steadily mounting diamond costs in the postwar years led to consideration of percussion drilling. Early trials were unsuccessful, because of inadequate equipment. Later, percussion drilling became feasible, with development of larger machines capable of handling larger steel, improved bit design, and improved coupling devices. The change to percussion drilling had little effect on the mining method but reduced the drilling cost one-third and increased fragmentation by permitting larger holes and closer spacing.

Fragmentation improved with experience through the years. At first a large tons-broken-per-foot-of-hole ratio was considered desirable, until it was found that the large tonnage factor was achieved at the expense of fragmentation. Large, unshattered slabs and boulders required excessive secondary blasting, with attendant damage to slusher drifts and drawholes. The latest trend was to smaller tonnage factors and better fragmentation. Tons broken per foot of hole ranged from 4 tons in 1947 to 8 tons in 1950. Later ratios ranged from 3.5 tons in wide stopes to less than 2.0 tons in narrow stopes (table 9).

Factors responsible for the lower tonnage ratios included the trend to smaller stopes in later years and a conscious effort to reduce caving and maximize fragmentation by closer spacing of blastholes and decreased burden. Millisecond-delay caps were important in gaining better control of blasting, with better fragmentation and less damage to stope walls. Reduced caving and less coarse muck resulted in fewer plugged drawholes, reduced secondary blasting, and increased slusher production. All

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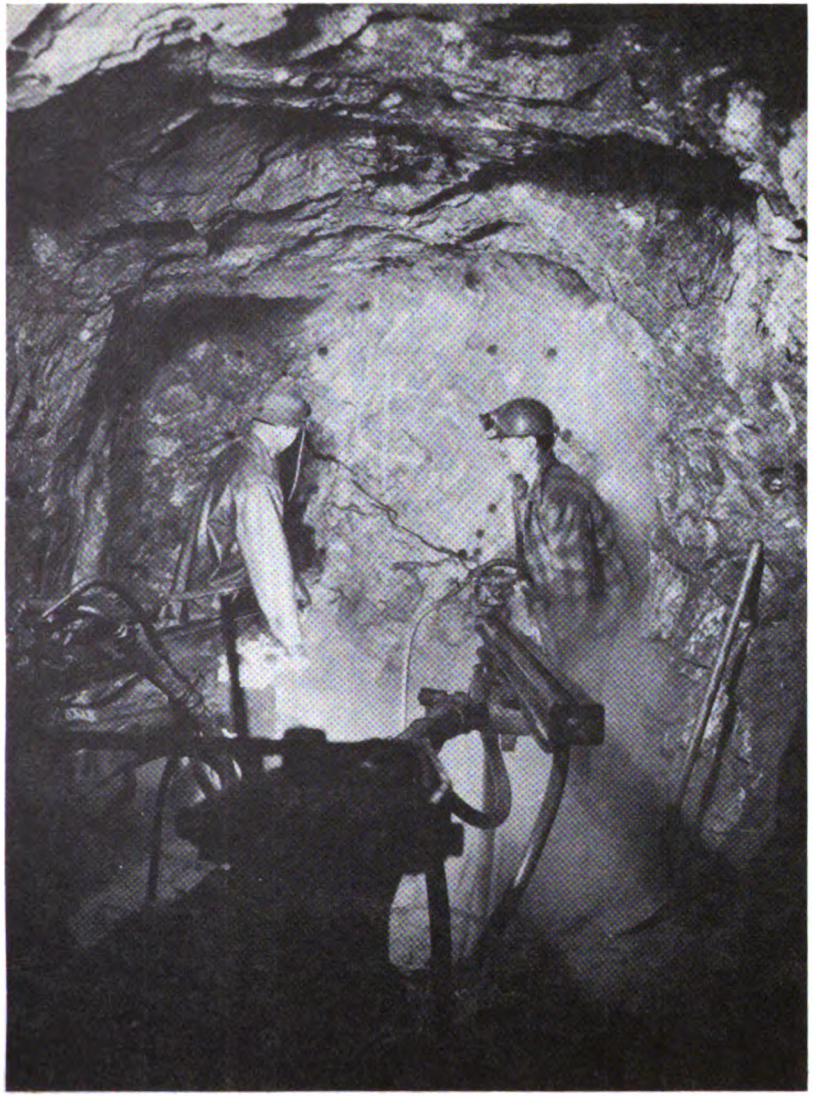

FIGURE 16. - Jumbo-Mounted Twin Drifters Drilling a Round in a Haulage Drift.

FIGURE 17. - Two-Man Crew Loading a 40-hole Burn-Cut Drift Round.

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these factors are interrelated; and, because some changes occurred concurrently, it is not possible to pinpoint their relative importance in increasing productivity.

Longhole Drilling

Three longhole ring patterns were used (fig. 18). The Holden ring is a fan-shaped pattern of flat holes, drilled from stations raised above the blasthole drifts. The holes, drilled from cramped quarters on staging, were difficult to drill and load. After a trial period, the Holden ring was abandoned, except for occasional use under special circumstances - for example, in a small stope whose size or location would not warrant more elaborate development or to avoid driving an extra blasthole drift to recover ore near the back of a stope.

The Noranda ring, used almost exclusively, is also fan shaped but is drilled in the vertical plane from blasthole drifts. It was preferred because it gave adequate coverage with no additional development; a number of rings could be drilled in advance of blasting, and there was no special hazard to the driller, as there was no need to work near the open stope.

Aldermac rings, seldom used except in very narrow stopes, consist of several long downholes drilled parallel to the stope face from an open-bench crosscut driven to the hanging wall from the blasthole drift. Disadvantage of the latter system are the hazards of working too near the open stope, alternate bench cutting and longholing, added expense of cutting a bench, and the necessity of separate drill setups for each hole drilled.

Placement of longholes received a great deal of consideration. The engineering department was responsible for designing and checking every ring drilled. Factors influencing the design were width of the stope, height of the ore block, degree of fragmentation desired, and natural features, such as fractures or faults, that might affect the blast. Longhole rings were designed so that no hole exceeded 65 feet in length. Beyond this point, penetration was slower, cuttings were difficult to remove, and efficiency dropped rapidly because of friction losses. Also, longer holes tended to wander and to become more susceptible to bootlegs.

Bootlegs (or rock remaining unbroken after being drilled and blasted) were minimized by avoiding, as much as possible, excessive burden to the free face and between holes, and drillholes normal to the hanging walls. Aldermac rings were preferred for narrow stopes for the latter reason. All holes paralleled the walls.

The outer end of the topmost hole of a ring was never higher than the back elevation of the blasthole drift. If it was desired to raise the back of the stope without resorting to raises and Holden rings, inclined holes that paralleled the strike of the ore body were drilled from crosscuts.

The maximum burden, or distance between rings, was 8-1/2 feet in stopes 60 or more feet in width and ranged down to 5 feet in narrow stopes 15 to 18 feet in width. The largest most recent stope was 60 feet in width; the majority was 20 to 25 feet in width and used rings with 6-foot burden. Within a single ring, the toe distance or burden between drillholes was especially critical for the two longest diagonal holes. This burden never exceeded 1-1/2 times the burden to the free face.

In general practice at least three longhole rings, or more if blasting schedules permitted, were drilled out ahead of blasting to maintain at least 15 feet between the driller and open stope. Even at that distance, a blast would often knock a drill down and cause a delay while it was set up again to resume drilling.

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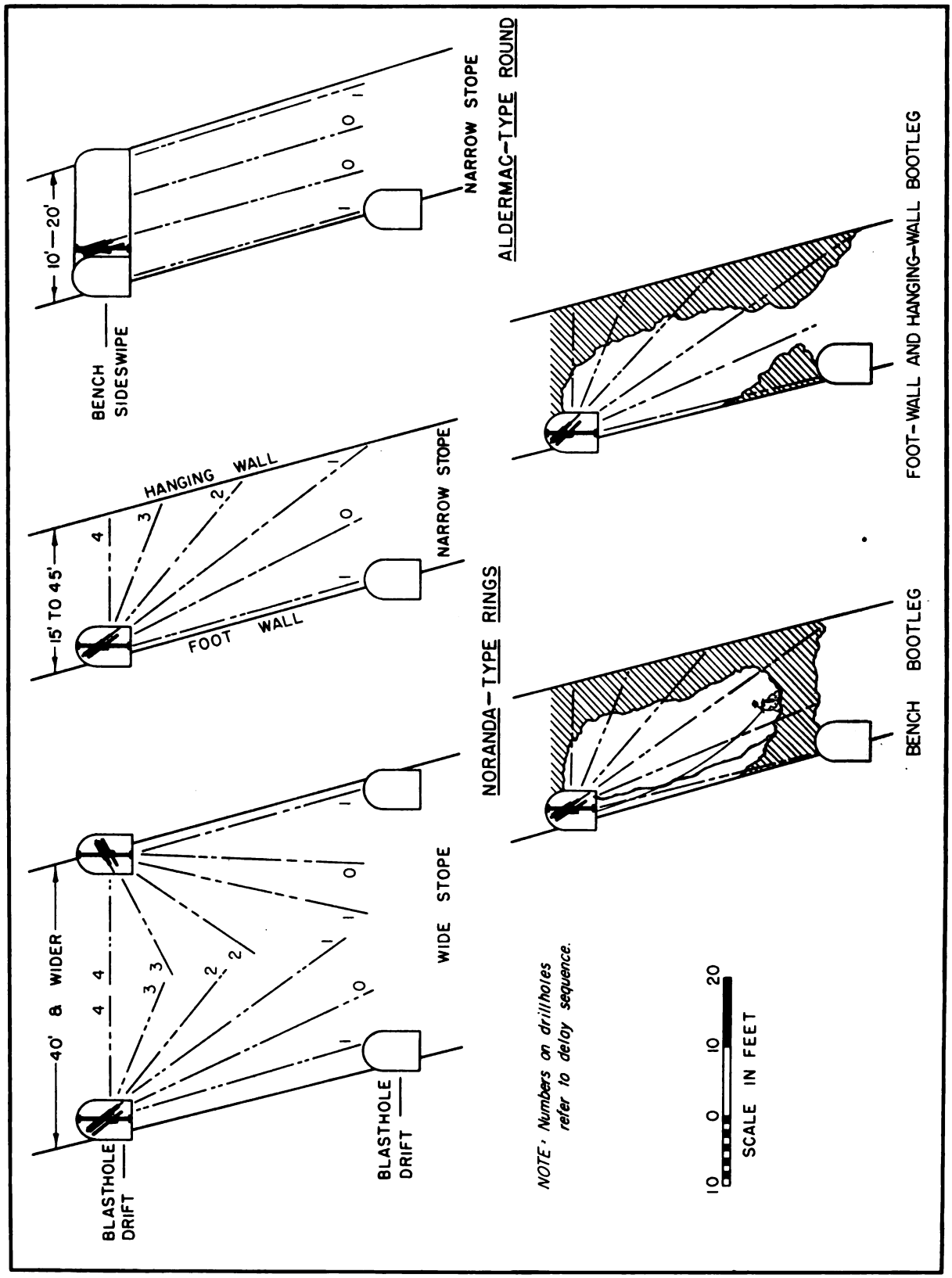

FIGURE 18. - Longhole Drill Patterns and Delay Sequence.

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Four-inch drifters with 36-inch power feed, mounted on 54-inch aluminum sliding-cone shells attached to a 3-inch vertical column, were used for longhole drilling (fig. 19). Two-inch carbide-insert bits were standard, although 2-1/2-inch bits were occasionally used for special purposes. Drill rods were 1-inch hexagonal alloy steel in 3- and 6-foot lengths. Longholes were collared in the conventional manner, and downholes were cased from solid rock to slightly above floor level to facilitate disposal of cuttings.

FIGURE 19. - Drilling Longholes With a Column-Mounted Percussion Drill.

Blasting

Blastholes were loaded with 60-percent dynamite in 12-inch preslit paper cartridges, 1-3/4 inches in diameter. Millisecond-delay caps in a series from 0 to 16 eliminated the possibility of cutoff holes. Typical delay patterns are shown in figure 15. Holes were loaded to within 2 to 7 feet of the collar, depending on their spacing, and were not stemmed. Primers were approximately 8 feet from the collar of the hole. A loading crew of 2 men could load 2,000 feet per shift under good conditions and normally averaged 500 to 600 feet.

Blasts involving several rings at once were commonplace. Shock waves from multiple blasts seemed to cancel out rather than be reinforced. Single-ring blasts were delayed so that the bottom or center holes were instantaneous, adjacent holes were

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primed with "1" delays to fire next, and so on in succession. In blasting multiple rings corresponding drillholes in the second and third rings were primed with longer delays, so that the first, second, and third rings were fired in that order. The same scheme of breaking from the center to the sides and from front to back was used, regardless of ring pattern.

Blasting often fractured the adjacent rock, so that previously drilled holes were offset or blocked. No holes were redrilled or reamed to size. The hole was loaded as completely as possible, and any bootleg that resulted was removed later.

Bench and hanging-wall bootlegs were common (fig. 18). Both resulted from excessive burden to the free face or excessive toe distance between holes in a ring. Foot-wall bootlegs seldom occurred, probably because the foot-wall drillhole paralleled the foot-wall plane. Hanging-wall bootlegs were removed by drilling long trimholes from the slash crosscut. Bench bootlegs that failed to be corrected by this treatment were drilled and blasted separately. This method was avoided if possible, as it involved lowering a man and drill into the open stope. Bootlegs must be removed, however, or the condition becomes steadily worse with succeeding blasts.

Secondary Blasting

The ore tended to break into large slabs and consequently hang up in the drawhole throats. Bombs were required to bring it down into the slusher drifts. Rocks too large to pass the grizzly were bombed or mudcapped to break them to size. Drilling was generally avoided because of the danger that loose powder from sloughed or cutoff holes might be in the muck pile.

Secondary powder consumption ranged from 0.85 to 1.30 pounds per ton in the last years (table 8).

Shrinkage Stopes

Longhole mining was by far the most important method used at Holden. From a total of 40 stopes below the 1500 level, only 2 were mined by shrinkage stoping. Both stopes, relatively narrow hanging-wall stopes, were mined by breasting down with airlegs and 10-foot steel. Setups were made on broken ore after enough ore was drawn to make room for drilling.

Handling Ore

Ore handling may be grouped into three distinct functions - ore gathering, crushing and hoisting, and main-level transportation to the mill. Alternate levels in the mine were designed for ore gathering; stopes on these levels had facilities for loading ore directly into cars, and stopes on intermediate levels had ore passes leading to chutes in the haulage level. In either instance ore was transported by train from gathering points to orepasses that led to crushers on the 1950 and 2650 levels. Crushed ore was hoisted through the main shaft to surge bins on the 1500 level, where it was transferred to ore trains for transportation to the mill.

Drawing Stopes

Ore was drawn from stopes to the slusher drift by gravity. On intermediate levels it was slushed to orepasses (fig. 20); on haulage levels it was loaded directly into cars by slusher or mucking machine.

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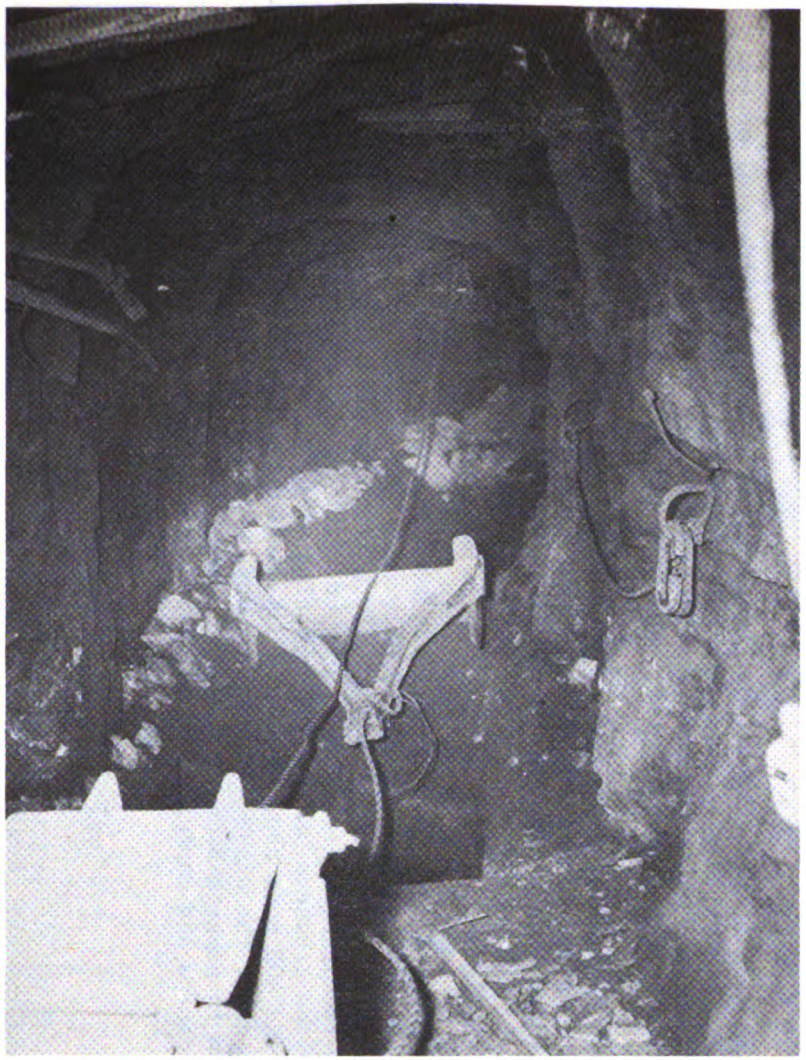

FIGURE 20. - Slusher Drift and Grizzly. White Mound in Background is a Drawhole Throat.

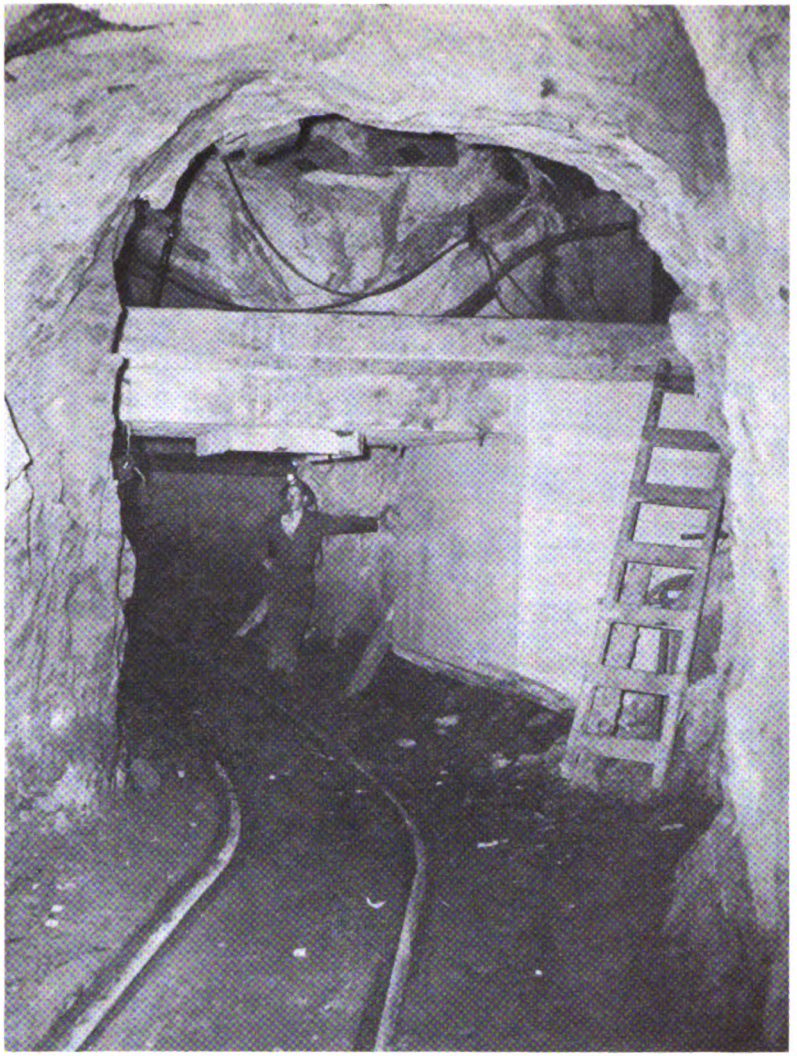

FIGURE 21. - Bathtub-Type Loading Station. Ore Cars Are Spotted Beneath Grizzly Opening To Be Loaded by Slusher. Slusher Drift Extends to Left of Concreted Grizzly Deck.

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Slusher hoists were 50-hp., double-drum units, geared to either 6,600- or 7,200-pound rope pull on the mainline. Slusher buckets were of patented design, developed in the company shops. The main features of the Clemmer bucket are interchangeable, hard-faced teeth and a unique pivoting drawbar that collapses on the back pull to permit easier passage through narrow openings. Mucking machines were the standard overcast type that load directly into cars.

The slushers were operated across grizzly bars set in concrete on 18-inch centers. From intermediate levels, the ore was dropped through to ore chutes on the haulage level below. For slushing directly into cars, a transfer station was established at the intersection of the slusher drift and a short crosscut to the main drift. The floor level of the slusher drift was 6 to 8 feet above the crosscut floor, and a concrete bridge spanned the gap (fig. 21). A grizzly opening in the bridge permitted cars on the main level to be loaded directly from the slusher drift.

Drawhole and Slusher Drift Repair

The quantity of ore that could be drawn from a drawhole varied with the type of rock and drawhole spacing. Drawholes on 20-foot centers and entirely within the ore zone sometimes required repair after passage of 30,000 to 40,000 tons of ore. Other drawholes on 30-foot centers and with the slusher drift in the footwall were still in good condition after 90,000 tons of ore had passed. The necessity for a large amount of secondary blasting in the slusher drifts was the biggest factor causing slusher-drift and drawhole damage. When drawhole throats became enlarged to an extent that the slusher drift was drowned in ore, the section was repaired with reinforced concrete and rock bolts. The remedy was only moderately successful, however, because the repairs were soon loosened or damaged by concussion.

Ore Chutes

Ore chutes were of heavy timber construction, with 1-inch, mild-steel bottom liners. Two gates were used, coarse muck was controlled by four downcast fingers, and fines were stopped by an undercut arc gate. Each gate was operated by independent air cylinders to give close control over draw (figs. 22 and 23).

A single experimental chute was constructed of concrete but was more expensive than timber. Concrete chutes can withstand abrasion and blasting better though, and with greater experience in design and construction might be competitive with timber.

Crushing and Hoisting

Primary crushing to minus 6 inches was accomplished underground by two 36- by 48-inch jaw crushers on the 1950 and 2600 levels. Production averaged 2,000 tons per 16 hours, operating 1 crusher at a time. Both crushers were powered by 125-hp. motors, fed by a drum feeder from air-actuated finger gates on the ore chutes. Coarse-ore pockets above the crushers had a total capacity of 2,000 tons, 1,600 tons above the 1950 crusher and 400 tons above the 2650. Additional coarse-ore storage was provided by ore chutes under the slusher drifts and ore passes above the crushers. Crushed ore was stored in pockets below the crushers and on the 1500 level in transit to the mill. The location and capacity of the various facilities are shown below.

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TABLE 1. - Ore-storage facilities

Location	Capacity, tons	Ore
1500-level ore pocket.....	1,500	Crushed.
Above 1950 crusher.....	1,600	Coarse.
Below 1950 crusher.....	1,100	Crushed.
2125-2500 ore pass.....	2,900	Coarse.
Above 2650 crusher.....	400	Coarse.
Below 2650 crusher.....	400	Crushed.
Total.....	7,900	

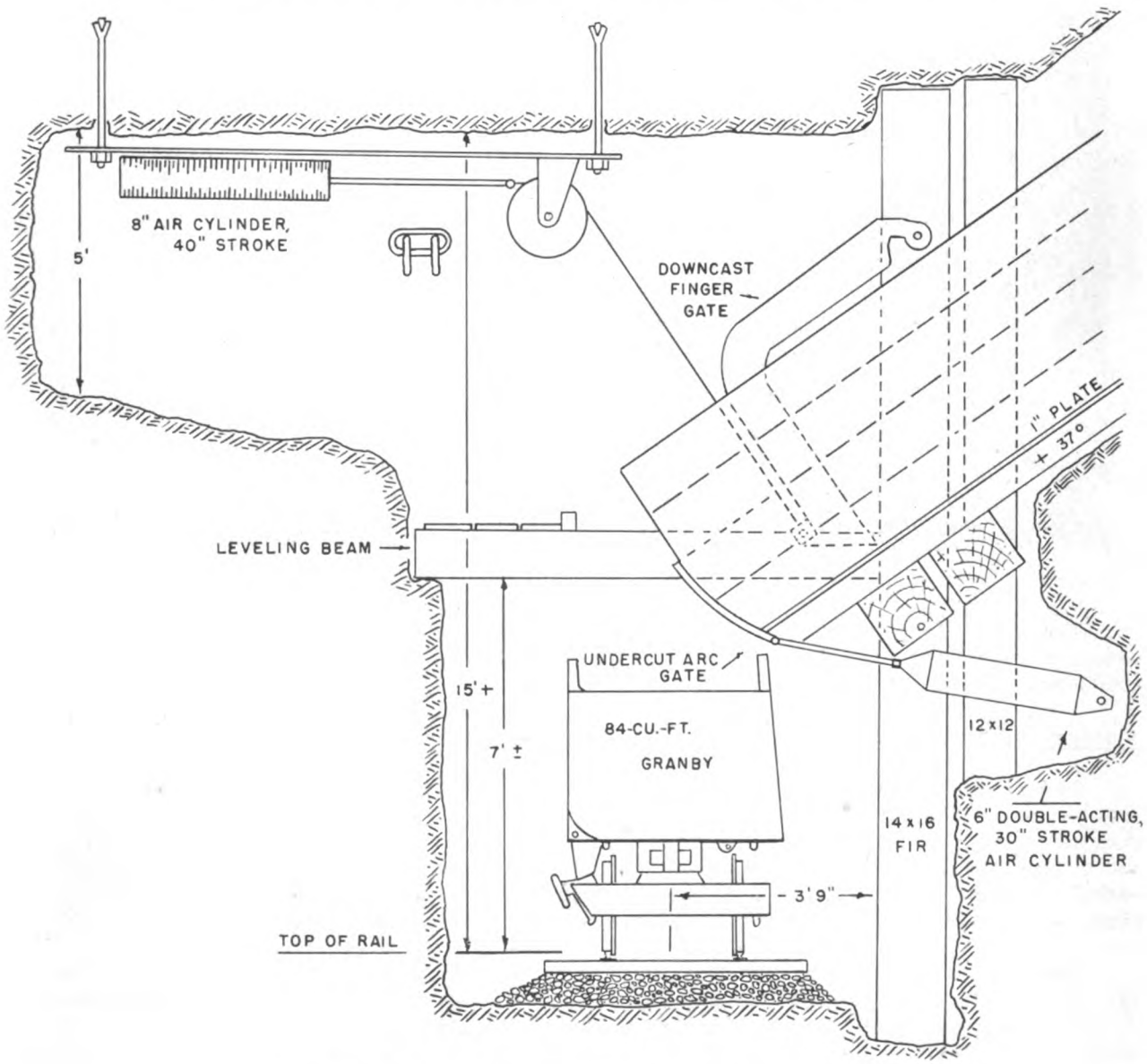

FIGURE 22. - Ore Chute and Air-Operated Chute Gates.

Actual storage was somewhat less than indicated, because 70 to 80 percent of the recent production went through the lower crusher; hence, storage facilities of the upper crusher were not fully utilized.

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FIGURE 23. - Loading Granby-Type Cars at an Ore Chute.

Ore was hoisted through the No. 2 shaft at the rate of 185 tons per hour by two 4-ton skips in balance (fig. 24). The ore hoist was a double-drum unit, powered by a 600-hp. motor, equipped with a Lilly controller.

Underground Transportation

Underground transportation was by rail. Direct-current trolley locomotives were used on ore-gathering levels and for main haulage transportation. Battery-powered trammers were used in exploration and development headings. The main haulage cars had a capacity of 170 cubic feet; ore cars used on the lower ore-gathering levels were 84 cubic feet in size. Both were Granby-type, side-dump cars. Rocker-dump cars, with 20-cubic-foot capacity, were used in smaller headings.

Two types of car dumps were used: The regular camelback-dump ramp was operated by a solenoid switch, which caused it to move into position as the loaded train passed the dump slot and to move out of position as the empty train returned (fig. 25). A manually operated, airpowered cable hoist was used where a less elaborate installation was desired - for example, at development-heading transfer stations. A stirrup, attached by cable to an air cylinder, was placed on the fifth wheel, and the cars were dumped separately by an operator (fig. 26).

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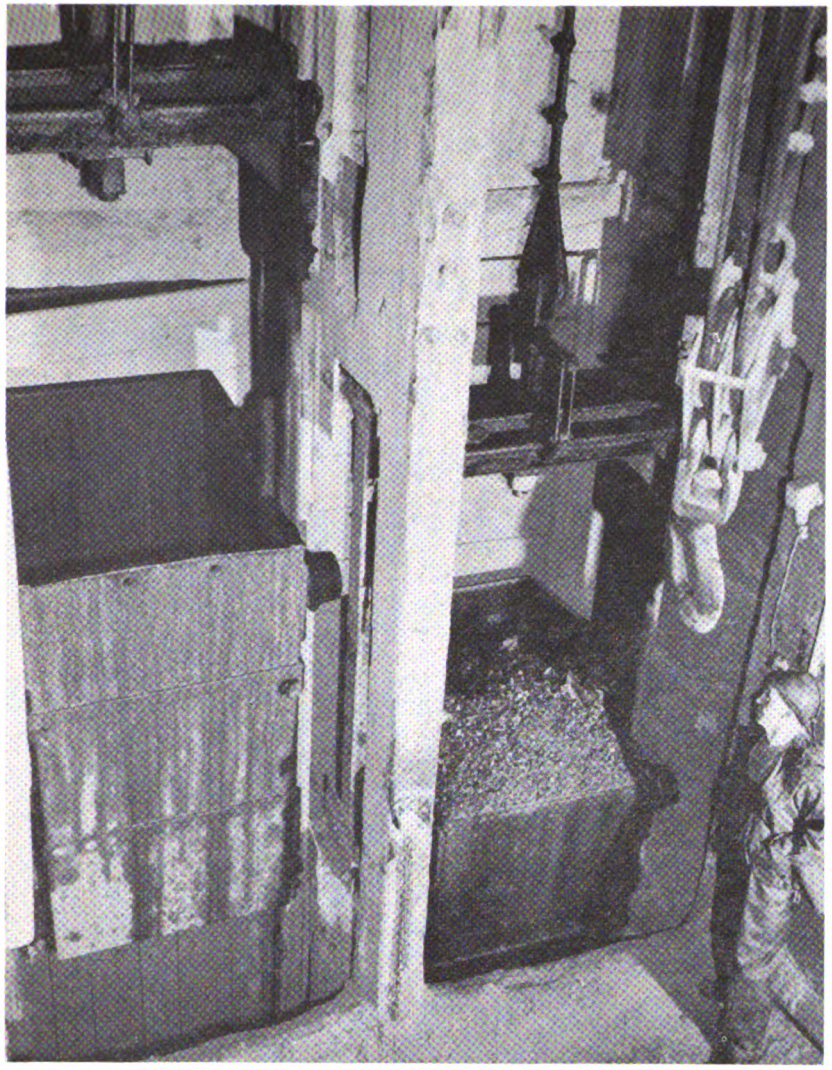

FIGURE 24. - Balanced Ore Skips in No. 2 Shaft Have 85-Cubic-Foot Capacity.

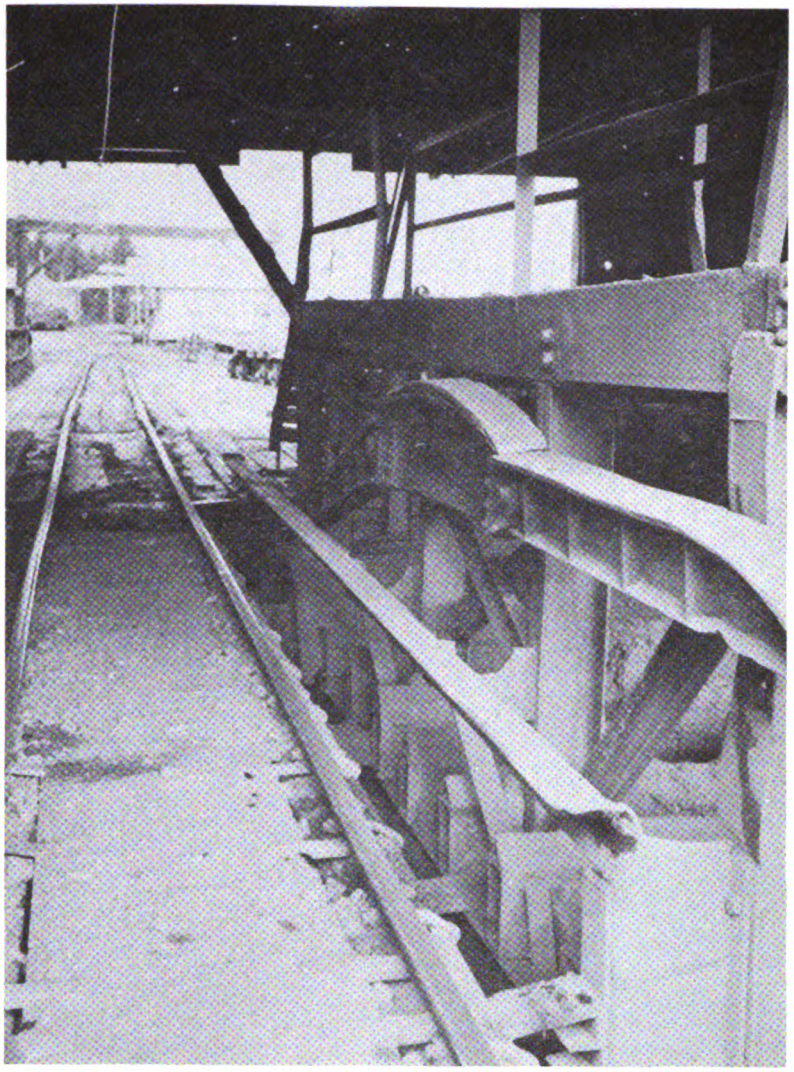

FIGURE 25. - Camelback Dump Ramp Similar to Those Used Underground.

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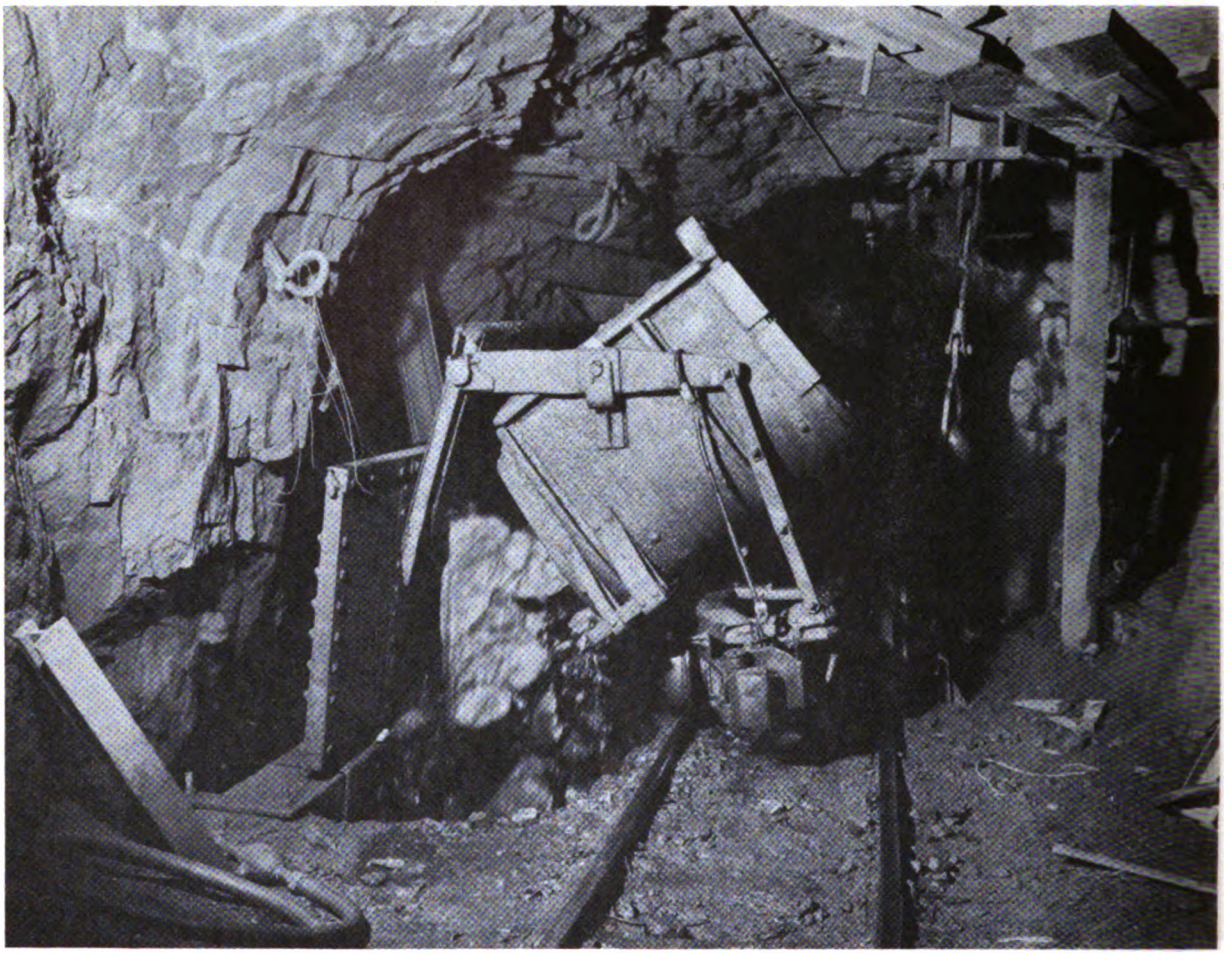

FIGURE 26. - Cable-Dump Device Used in Subsidiary Headings.

Ground Control

Few problems in ground support were encountered in the mine above the 1500 level. The tremendous empty stopes in this area are in nearly the same condition today as when they were abandoned (fig. 27). Some of these stopes were carried to within 50 feet of the surface, and yet none of the cappings have caved.

As stopes were opened below the 1500 level, ground conditions became more unstable. This was alleviated somewhat by filling abandoned stopes with mill tailing, by a more orderly sequence of stoping, and by mining smaller stopes. Universal support was not necessary in the lower workings; in fact, sizable sections lacked artificial support of any kind. In other parts of the mine squeezing ground and spalling rock were troublesome. Slusher drifts, especially those in weak ground, were troublesome because of repeated blasting. Besides sand filling, other methods of support used were various combinations of rock bolts and headboards, conventional timbering, caps and pins, reinforced concrete, and gunited wire mesh.

In recent years conventional timbering was used little for permanent support, except to supplement rock bolting. Timber caps, supported by steel pins set in the ribs and covered with lagging, largely supplanted conventional timbering for back

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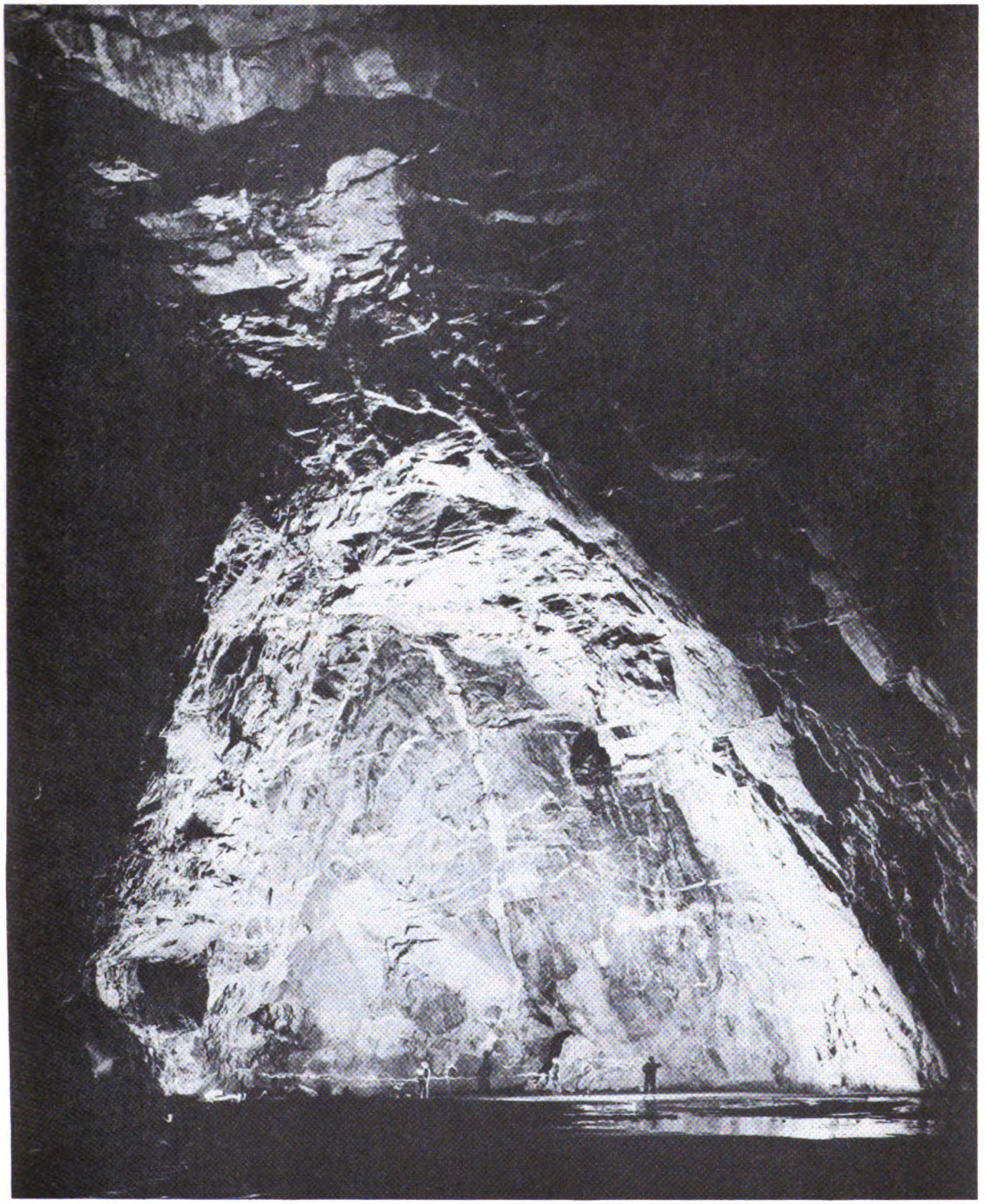

FIGURE 27. - Stope 257 Above 1500 Level. Only Portion of Stope Above Main Level Remains Open; Remainder is Sand-Filled. Note Foot- and Hanging-Wall Drift.

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support. In areas where side pressures developed, posts and side lagging were added for additional protection.

Rock Bolting

Rock bolts were used extensively in patterns and at random. Wedge-type bolts with burned slots were manufactured in company shops, and expansion-shell types were purchased. In active ground the biggest problem is to provide enough bearing surface to give maximum support. Bolts may remain tight although rock between the bolts spalls away (fig. 28). Spalling is not critically dangerous, but the rock shards have enough energy to cause painful injuries, and eventually the bolts become ineffective. Various types of headboards, caps, and plates were used; the most successful were 8-inch steel plates with beveled washers, and 3-inch timber headboards pulled tight and blocked with wedges. The first is less susceptible to damage from blasting and therefore is used more often in slusher drifts and near advancing faces. Timber headboards have more bearing surface and are preferred where that is a consideration. Channel iron, discarded rail, and other materials have been used for headers with indifferent success, because it is difficult to make rigid material conform to an irregular surface and still obtain enough tension on the rock bolt.

Grouted rock bolts were tried in slusher drifts, but the grouting procedure was so tedious and time consuming that the results were not considered worthwhile.

Gunited wire mesh on rock bolts and reinforced concrete were used in slusher drifts, but repeated concussion and rock pressure eventually caused failure.

Sand Filling

The practice of using deslimed mill tailing for stope fill was begun in 1947. Ground conditions became increasingly hazardous as mining depth increased and more open stopes were developed. From 1947 to 1954, when sand filling was discontinued, nearly 1.8 million tons of mill tailing was placed in stopes below the 1500 level.

Sand fill was obtained from two sources - directly from the mill circuit and from previously impounded sand in the tailing pond. The latter was loosened by a hydraulic giant, passed through a grizzly to remove debris and oversize material, and pumped to 1 of 3 deslimers at the mill. After desliming and thickening to 40 to 50 percent solids, the slurry was pumped underground in 6-inch wood-stave pipe, dropped from the 1500 to lower levels through 6-inch rubber-lined pipe, and distributed to point of use in either 4- or 6-inch pipe. Sand filling was begun as soon as possible after stoping was completed and continued, except for mechanical interruptions, until the stope was filled.

Stope preparation consisted of erecting bulkheads on all stope openings and sealing cracks and other rock defects with cement. It was originally intended that excess water would be pumped out or decanted at available horizons as the stope filled. Decantation was satisfactory for short periods before the opening was overrun with sand. Pumping from the overlying pool of water was overly expensive and required frequent pump handling. In practice it was found that most of the drainage occurred through minor fractures in the footwall and around the bulkheads. The only disadvantage was that drainage was not under control, and considerable sand escaped into passageways before leaks sanded up to pass only clear water. Drainpipes were later installed in the bulkheads to provide a measure of control over drainage (fig. 29).

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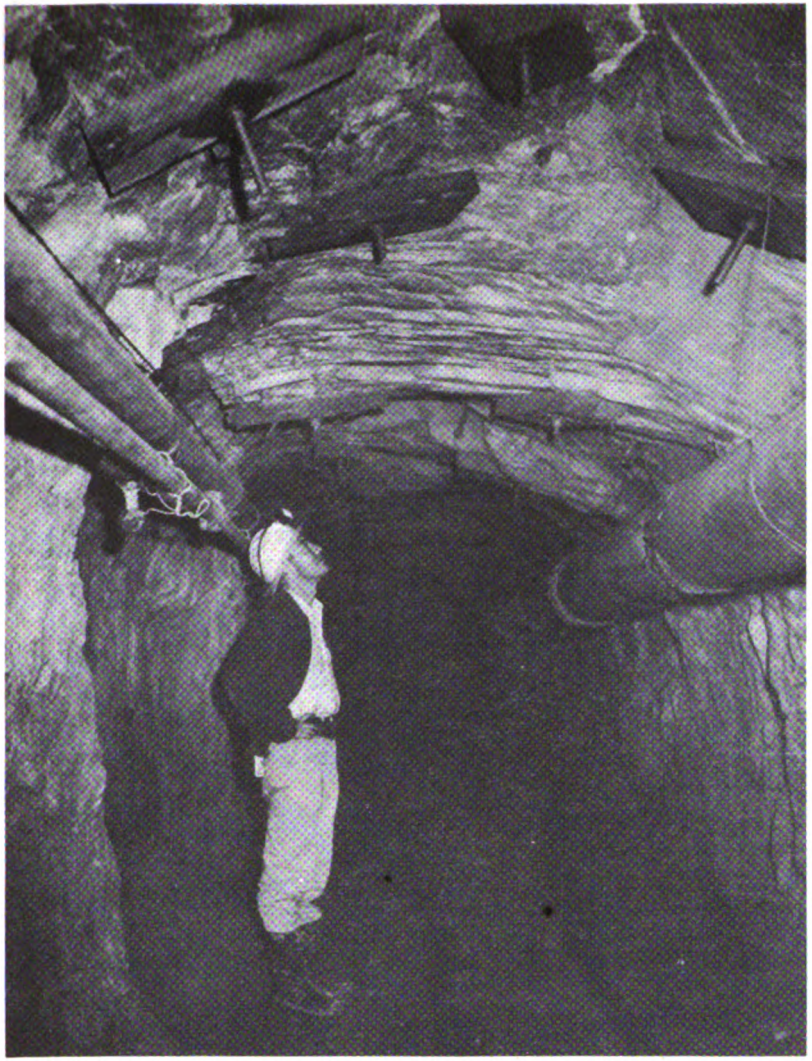

FIGURE 28. - Rock Bolts in an Exploration Drift Remain Tight While Intervening Rock Spalls Away to Form a Natural Arch.

FIGURE 29. - Concrete Bulkhead in a Sand-Filled Stope. Note Drainpipe in Upper Right Corner. Failure of a Similar Bulkhead Flooded Mine in 1951.

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The only serious incident relating to sand filling was caused, at least to a large extent, by too little drainage. Previously, all stopes being filled had considerable natural drainage. In the 257 stope holes drilled originally to sample the stope provided the only outlet for water; natural drainage was negligible. On April 4, 1951, one of the concrete stope-fill bulkheads ruptured, flooding the lower levels with 8,100 tons of sand and 2,186,000 gallons of water. The rupture, approximately 4 by 12 inches, was attributed to several possible causes, including defective bulkhead construction due to poor concrete work or fracturing of the bulkhead by ground movement. Either condition may have been aggravated by excessive hydrostatic pressure, concussion from nearby blasting, or a water-hammer effect, caused by concussion in an adjacent stope acting on a water-filled connecting raise. In any event inadequate drainage had permitted the sand fill to remain in a semifluid state. When mining was resumed after the cleanup, adequate stope drainage received greater attention.

Sand filling was discontinued in 1954 when some doubt arose as to its value as a ground-stabilization measure. Pertinent data relating to the filling of a typical stope are summarized in table 14.

VENTILATION

Both natural and mechanical ventilation was used. In the summer months fresh air was drawn in through the 1500 level. During the winter the inflow of air on this level was restricted by an automatic double-door airlock to prevent the level from freezing. Air was then drawn from the 550 and 800 levels through old stopes and ventilation raises. Airflow generally was eastward on the main level to a 50,000-c.f.m. booster fan at the east end of the mine, downward through a series of downcast fresh-air raises to the 2500 level, and westward through the mine workings to upcast exhaust raises at the west end of the mine. A 110,000-c.f.m. fan on the 1500 level discharged exhaust air through a ventilation adit a short distance west of the main portal. Fresh air was carried through haulage levels to slusher drifts by 6-foot ventilation raises, driven, where possible, in end pillars to serve two adjacent stopes. Each slusher unit received a minimum of 5,000 cubic feet of air per minute. Fresh air flowed past the slusher operator, grizzly, and drawholes, in that order, and exhausted into upcast raises at the far end of a slusher drift. Exhaust air collected in intermediate levels and traveled west to the main upcast exhaust raise. Ventilation was controlled by doors at strategic points in the passageways and ventilation raises. The main downcast-air raise served also as an escapeway. Exploration and development-heading ventilation was provided by 15-hp. fans through 15-inch, 16-gage, black steel pipes.

DRAINAGE

The mine was relatively dry, and drainage was not a problem. Seepage that collected at the shaft bottom and in sumps on the 2125 level was pumped to the surface in two stages. Pump specifications are shown in table 15.

SAFETY

A full-time safety engineer was employed to supervise maintenance of safe working conditions. Periodic safety inspections of the entire mine were made by a safety committee representing both labor and management. The mine foreman was responsible for carrying out recommendations of the safety committee. Until the last year, Bureau of Mines first-aid instruction was given to all interested persons. First-aid kits and stretchers were easily accessible at shaft stations, and an ambulance was maintained in a heated garage at the 1500-level portal. Complete hospital facilities with

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a doctor and nurse in attendance were maintained by the company at the townsite. Pre-employment physical examinations were required of all miners. The success of the safety program is indicated in the following table:

TABLE 2. - Accident rate

	1956	1955	1954	1953	1952
Compensable accidents.....	3	0	2	2	2
Accidents per 100 man-shifts.....	0.528	0.000	0.324	0.302	0.380
	1951	1950	1949	1948	1947
Compensable accidents.....	0	1	1	2	1
Accidents per 100 man-shifts.....	1.394	0.471	0.654	0.615	1.657

Powder was stored in a well-identified, revetment-type magazine about 500 yards from the main portal; cap storage was underground on the 1500 level. Underground powder magazines were on the 2500, 2100, and 1700 levels. Primers were made on the 1400 level. All magazines were identified, and customary signs were displayed prominently.

The mine never had an underground fire. Smoking was prohibited at the shaft stations, and the shafts were kept continually wet. Concrete and steel fire doors were installed in all shaft stations to confine and smother any fire that might start, and a fire hose was kept at the 1500 shaft station, ready for any emergency.

LABOR

As mentioned before, the mine is in an isolated locality. To attract and retain competent workmen, the company endeavored to create and maintain attractive working and living conditions. Consequently, Holden was one of the most pleasant and modern mining camps in the West. More than half the employees were permanent; that is, they had been with the company for a number of years. About 30 families were at Holden from its beginning. Another quarter of the force was semipermanent, and the remainder itinerant. As a result, labor relations generally were good. Wage scales and benefits were patterned after those prevailing in the Coeur d'Alene, Metaline, Butte, and other Northwest mining districts. All employees, excluding supervisory, administrative, and engineering personnel, were represented by Local Union 379 of the International Union of Mine, Mill, and Smelter Workers.

WAGE AND BONUS SYSTEMS

Typical day's-pay wage scales are shown in table 6. A bonus system, calculated to enable a good worker to earn a bonus above day's pay, according to his ability, was in existence since 1942. Originally designed to overcome the wartime manpower shortage, the system was retained and expanded to cover most underground operations. Bonus prices were computed on the basis of normal output for each phase of the operation; that is, a suitable price was paid for each unit of production, such as foot of advance, linear feet of timbering, tons of ore drawn and hauled, etc. In establishing bonuswork the company specified the size of heading or other unit of production, the number of men on the bonus crew, the quality of work, and the services that would be provided either by the bonus crew or by the company. Bonus work was measured weekly. Bonus crews producing exactly the normal output received what amounted to day's pay; production in excess of the normal was paid for at the bonus price. No crew received less than day's pay. Bonus prices may have varied slightly, depending on services provided, length of tram, and mucking equipment used. Prices were the same, regardless of whether the work was in waste or ore.

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APPENDIX

TABLE 3. - General-cost summary[1/]

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Dry tons milled	309,435	398,738	444,694	433,717	545,776	550,530	657,634	627,316	232,158
Mining expense:									
Direct	53.3	51.5	54.8	51.5	46.4	46.9	44.7	44.8	44.5
Development	2.6	3.1	3.9	6.8	5.8	6.7	7.0	7.7	9.2
Core drilling	.5	.2	.6	.9	1.5	1.4	2.1	1.7	1.5
Ore transportation (underground)	1.4	1.3	1.4	1.5	.8	.6	1.0	1.7	1.4
Milling expense	18.4	20.0	18.2	18.2	24.3	24.5	26.9	25.3	24.6
Surface transportation (to railhead)	5.0	5.5	4.5	4.9	5.2	4.9	4.8	4.9	4.2
Administration	4.5	4.5	4.1	4.0	4.2	4.0	3.7	3.8	4.1
Property tax[2/]	1.2	1.1	1.0	1.0	1.0	.9	1.0	.9	.9
Other	13.1	12.8	11.5	11.2	10.8	10.1	8.8	9.2	9.6
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Comparative expense[3/]	156.2	126.5	123.3	126.9	105.4	103.7	95.7	93.9	100.0

1/ Percentage of total cost.
2/ Excluding income tax and depreciation.
3/ Base year 1948.

TABLE 4. - Mining-expense summary[1/]

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Tons broken	253,531	425,294	349,822	381,515	479,894	501,404	393,991	452,617	416,327
Tons drawn from stopes	283,812	374,553	407,268	398,041	520,666	521,515	614,399	594,289	587,559
Tons drawn from development	23,735	25,734	38,564	38,629	24,316	29,691	39,054	35,601	22,165
Total tons drawn	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Ore breaking, direct expense:									
Labor	8.59	7.81	9.70	9.19	4.75	6.02	6.52	7.22	6.37
Drilling and blasting	13.93	12.86	12.89	11.39	7.49	7.19	7.08	10.92	14.70
Explosives	4.82	4.57	4.24	3.71	2.36	2.77	2.96	3.67	3.08
Timber	1.24	.06	.06	.16	.06	.08	.51	.44	.33
Other	3.29	.23	.23	.30	.17	.11	.13	.04	-
Total	31.87	25.53	27.13	24.75	14.82	16.17	17.20	22.28	24.48
Ore drawing:									
Labor	9.45	11.13	10.23	11.00	9.63	10.20	9.78	9.64	8.70
Explosives	4.78	5.76	6.89	6.88	6.63	9.69	10.97	10.26	9.08
Other	2.72	2.47	1.73	2.46	1.95	3.20	3.14	2.48	2.29
Total	16.95	19.36	18.85	20.34	18.21	23.09	23.89	22.38	20.07
Total ore breaking	31.87	25.53	27.12	24.75	14.83	16.17	17.20	22.29	24.48
Total ore drawing	16.95	19.36	18.85	20.34	18.21	23.09	23.89	22.38	20.07
Tramming	11.38	11.38	11.55	11.52	7.78	8.55	8.47	8.54	7.66
Hoisting	7.50	7.36	7.01	6.77	7.02	7.15	6.78	6.50	7.24
Mine general expense	26.71	30.16	25.39	21.93	19.13	21.60	21.35	21.14	21.02
Development	5.59	6.21	10.08	14.69	13.83	17.93	22.31	19.15	19.53
Other	-	-	-	-	19.20	5.51	-	-	-
Total mining expense	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Comparative expense[2/]	168.32	129.43	139.51	160.32	130.56	106.58	98.29	94.13	100.00

1/ Percentage of total expense.
2/ Compared with base year 1948.

TABLE 5. - Labor distribution and production, January 1 to December 31, 1955

		Man-shifts per day			Man-hours per day			Man-hour per ton		
		Production	Development	All	Production	Development	All	Production	Development	All
Breaking		8.9	14.1	23.0	71.2	112.8	184.0	0.0540	0.0856	0.1396
Timbering		8.8	2.3	11.1	70.4	18.4	88.8	.0534	.0140	.0674
Mucking		14.4	11.5	25.9	115.2	92.0	207.2	.0874	.0698	.1572
Hauling and hoisting		17.6	6.3	23.9	140.8	50.4	191.2	.1068	.0382	.1451
Supervision		3.3	3.4	6.7	26.4	27.2	53.6	.0200	.0206	.0406
Crushing		1.8	-	1.8	14.4	-	14.4	.0109	-	.0109
Diamond drilling		-	2.2	2.2	-	17.6	17.6	-	.0134	.0134
Maintenance		-	-	13.2	-	-	105.6	-	-	.0801
Total underground labor		54.8	39.8	107.8	438.4	318.4	862.4	.3326	.2416	.6543
Chargeable surface labor		-	-	43.0	-	-	344.0	-	-	.2609
Total labor				150.8			1,206.4			.9153
Tons broken	425,294									
Tons drawn	400,287									
Work days	303.7									
Avg. daily production	1,318.03									
Avg. daily underground shifts	107.8									
Avg. chargeable surface shifts	43.0									
Tons per man-shift:										
Underground		12.23								
Surface		30.65								
Overall production		8.74								

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TABLE 6. - Job classifications and wage scales

Job classifications			Wage scales	
Mine	Shops	Surface	Effective Feb. 1, 1957 1/	Effective July 1, 1957
Shaft miner (senior)	Mechanic (senior), electrician (senior)	-	$19.87	$20.55
-	-	Power-shovel operator	19.41	20.07
Shaft miner, timber framer, diamond driller, hoistman	Welder, mechanic, blacksmith, electrician, machinist	Heavy-equipment operator, mechanic, carpenter, painter, plumber, first cook	18.95	19.59
Longhole loader	-	-	18.49	19.11
Miner (raise, drift, stope), longhole driller, timberman	Bit sharpener	Heavy-truck driver	18.03	18.63
Mucking-machine operator (production), motorman (main level), slusher operator (production), trackman	Welder, mechanic electrician, blacksmith (second class), machinist	Mechanic, carpenter (second class)	17.57	18.15
Cage or skip tender, pipeman, mucking machine operator (development)	-	Baker	17.11	17.67
Slusher operator (development), fuseman, motorman (sublevel), brakeman, powder nipper	Lampman	Truckdriver, hooktender, oiler	16.65	17.19
Helpers, mucker, nipper	Helpers	Second cook	16.19	16.71
-	-	Watchman	15.73	16.23
-	-	Surface labor, dishwasher, waiter, janitor	15.27	15.75

1/ Retroactive to July 1, 1956.

TABLE 7. - Distribution of electric power 1/

	1956	1955	1954	1953	1952	1951	1950	1949	1948
Dry tons milled	309,435	398,738	444,694	433,717	545,776	550,530	657,634	627,316	608,431
Mine:									
Compressors	10.38	9.14	8.83	9.14	7.33	7.79	7.05	6.56	7.57
Hoists	2.34	2.07	2.07	2.13	1.92	2.10	1.98	1.99	1.51
Slushers	.35	.29	.24	.21	.23	.72	1.23	.91	.50
Pumps	2.23	2.04	2.59	2.69	2.51	1.94	1.79	.95	.74
Ventilation	4.65	3.23	2.33	2.53	2.12	2.36	2.17	1.80	1.99
Crusher	.50	1.30	1.06	1.92	1.54	.70	.38	.23	.17
Tramming	1.02	.83	.71	.86	.72	.67	.53	.64	.67
Stope fill	-	-	.32	.70	1.29	.59	1.17	.74	-
Lighting and other	.61	.47	.43	.41	.23	.18	.13	.18	.14
Total	22.08	19.37	18.58	20.59	17.89	17.05	16.43	14.00	13.29
Mill:									
Coarse crushing	2.08	2.55	2.74	3.03	2.85	2.73	2.69	2.64	3.06
Fine grinding	12.57	12.04	11.79	11.49	12.97	12.50	12.19	12.53	13.36
Flotation	6.21	5.39	5.21	5.84	5.57	5.03	4.11	4.46	5.13
Other	1.92	1.77	1.78	2.48	1.68	1.71	2.97	4.38	3.84
Total	22.78	21.75	21.52	22.84	23.07	21.97	22.68	24.01	25.39
General, surface	8.77	6.60	5.99	5.82	4.83	4.30	3.55	3.32	3.47
Total	53.63	47.72	46.09	49.25	45.79	43.32	42.66	41.33	42.15

1/ Kw.-hr. per dry ton milled.

TABLE 8. - Powder consumption

	To date	1956	1955	1954	1953	1952	1951	1950	1949	1948
Ore broken:										
By longholes	4,846,562	193,367	394,211	303,037	328,562	452,959	462,715	341,215	409,947	380,169
From other sources	3,700,089	60,164	31,083	46,775	52,953	26,935	58,800	52,776	42,670	36,153
Total	8,546,651	253,531	425,294	349,822	381,515	479,894	521,515	393,991	452,617	416,322
Ore drawn:										
From stopes	-	283,812	374,553	407,268	398,041	520,666	521,515	614,399	594,289	587,559
From development	-	23,735	25,734	38,564	38,629	24,316	26,691	39,054	35,601	22,165
Total	10,188,383	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Ore broken, tons	8,546,651	253,531	425,294	349,822	381,515	479,894	521,515	393,991	452,617	416,322
Ore drawn, do.	1/ 10,188,383	307,547	400,287	445,832	436,670	544,982	551,206	653,453	629,890	609,724
Apparent dilution, percent	19.5	21.3	-5.9	27.4	14.5	13.6	5.7	65.9	39.17	46.5
Mine development, feet	-	1,262	1,315	2,298	3,827	2,795	3,814	4,835	5,611	6,077
Stope development, do.	-	5,709	5,600	8,023	7,316	3,844	4,816	7,006	8,071	4,607
Exploration, do.	-	300	3,846	1,086	1,017	1,144	2,344	2,308	-	-
Total, do.	-	7,271	10,761	11,407	12,160	7,783	10,974	14,149	13,682	10,684
Primary breaking powder factor 2/	0.23	0.45	0.33	0.32	0.28	0.18	0.14	0.18	0.18	0.26
Secondary breaking powder factor 3/	-	0.85	0.84	1.03	1.29	1.13	1.30	(4)	(4)	(4)
Total powder for production	-	1.16	1.19	1.26	1.51	1.29	1.42	-	-	-
Powder per foot of development	-	29.3	22.3	25.63	23.9	23.8	27.4	(4)	(4)	(4)
Total powder consumption per ton production	-	1.76	1.71	1.81	2.04	1.57	1.89	2.12	1.88	(4)

1/ Plus 24,985 tons reserve.
2/ Pound per ton broken.
3/ Pounds per ton drawn.
4/ Not available.

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TABLE 9. - Blasthole performance

	To date	1956	1955	1954	1953	1952	1951	1950	1949	1948
Drilled.....feet	1,362,315	83,588	130,831	106,296	75,205	65,721	49,657	48,112	84,638	71,889
Blasted.....do.	1,292,798	83,745	133,807	85,243	70,415	76,091	52,289	48,963	68,290	82,777
Broken.....tons	4,846,562	193,367	394,211	303,037	328,562	452,959	462,715	341,215	409,947	380,169
Blasted.....tons per foot	3.75	2.31	2.95	3.53	4.67	5.95	8.85	6.97	6.00	4.59
Powder.....pounds	1,099,871	87,600	131,550	95,090	87,750	83,276	64,100	62,375	75,575	98,465
Powder.....,pounds per ton	.23	.45	.33	.32	.28	.18	.14	.18	.18	.26

TABLE 10. - Apportionment of longhole drilling costs

	Since inception	1956	1955
Ore breaking.....feet	664,169	83,588	130,831
Development.....do.	60,587	4,148	6,412
Total drilled.....do.	726,493	87,736	137,243
Bits used.....	3,835	337	452
Feet per bit.....	189.4	260.3	303.6
Direct expense:1/			
Labor.....	58.65	60.91	64.20
New bits.....	14.29	11.27	11.78
Grinding.....	2.44	2.72	2.77
Couplings.....	3.20	2.18	3.23
New drills.....	3.76	-	2.54
New rods.....	5.64	6.92	4.16
Rod maintenance.....	1.69	.55	.93
Drill repair.....	5.08	5.45	5.77
Other labor and supplies.....	5.26	10.00	4.62
Total.....	100.00	100.00	100.00
	2/72.98	2/76.39	2/72.90
Air.....	14.13	13.89	14.14
Air and water lines.....	12.89	9.72	12.96
Total.....	100.00	100.00	100.00
Total drilling expense3/.....	(4)	17.82	19.10

1/ Percentage of direct drilling expense.
2/ Percentage of total drilling expense.
3/ Percentage of total mining expense.
4/ Not available.

TABLE 11. - Apportionment of costs1/ and comparison of ore-drawing methods

	Slushing to ore passes			Slushing to cars			Mucking into cars		
	1956	1955	1954	1956	1955	1954	1956	1955	1954
Ore drawn.....tons	118,103	325,573	201,570	41,349	-	33,701	33,915	68,638	67,766
Labor.....	32.26	37.43	37.24	42.82	-	56.35	53.71	61.40	56.81
Supplies.....	2.45	1.23	1.40	.14	-	-	-	-	-
Explosives.....	28.44	31.81	41.60	25.18	-	31.55	29.71	22.40	21.48
Power.....	.46	.35	.35	-	-	.40	-	-	-
Maintenance:									
Mechanical.....	16.67	16.17	16.26	5.97	-	11.70	-	-	-
General.....	19.72	13.01	3.15	25.89	-	-	-	-	-
Other.....	-	-	-	-	-	-	16.58	16.20	21.71
Total.....	100.00	100.00	100.00	100.00	-	100.00	100.00	100.00	100.00
Cost of ore drawing2/.....	16.18	18.30	17.06	17.40	-	15.04	22.99	23.83	25.42
Tons per hour per unit.....	34.6	35.8	(3)	22.4	-	(3)	15.5	14.0	12.2

1/ Percentage of ore-drawing expense.
2/ Percentage of mining expense.
3/ Not available.

TABLE 12. - Apportionment of transportation costs, 1500 level

	1956	1955	1954
Ore hauled.....tons	309,657	398,556	444,808
Cars.....	39,312	51,943	58,736
Tons per car (dry).....percent	7.88	7.81	7.57
Direct operating costs:1/			
Labor.....	37.17	44.21	45.92
Supplies.....	-	-	.14
Power:			
Air.....	4.09	5.00	5.12
Electric.....	2.32	2.50	2.49
Total direct operating costs.....	43.58	51.71	53.67
Track and rolling stock maintenance.....	56.42	48.29	46.33
Total, 1500 level transportation costs.....	100.00	100.00	100.00
Cost of 1500 level transportation2/.....	19.65	21.47	18.69
Cost of 1500 level transportation3/.....	2.24	2.44	2.16
All tramming costs as percentage of mine expense.....	11.38	11.38	11.55

1/ Percentage of transportation costs, 1,500 level.
2/ Percentage of total tramming cost.
3/ Percentage of total mine expense.

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TABLE 13. - Apportionment of hoisting expenses[1/]

	1956	1955	1954
Ore hoisted.....tons	307,547	400,287	445,832
Waste hoisted.....do.	4,207	7,550	13,390
Direct operating expenses:[2/]			
Labor.....	65.00	64.26	56.04
Supplies.....	1.39	.33	.33
Power.....	5.39	5.62	5.53
Air.....	1.36	1.92	1.73
Total direct expense.....	73.14	72.13	63.63
Hoist and shaft maintenance.....	26.86	27.87	36.37
Total hoisting expense.....	100.00	100.00	100.00
Percentage of total mine expense.....	6.93	6.86	6.37
Apportionment:			
Man hoist.....	44.70	42.01	43.21
Ore hoist.....	55.30	57.99	56.79

1/ Main shaft only.
2/ Percentage of shaft hoisting expense.

TABLE 14. - Sand-fill statistics for 280 stope 1775 level, January-December 1947

Maximum depth of fill.....vertical feet	227
Average width of stope.....feet	55
Average length of stope.....do.	280
Average horizontal exposure.....square feet	15,000
Estimated volume to be filled.....cubic feet	3,105,000
Estimated weight of fill.....tons	165,500
Actual weight of fill.....do.	150,096
Specific gravity of mine rock.....cubic feet per ton	10.3
Specific gravity of fill (15 percent retained moisture).....do.	18.87

Typical screen analyses

Mesh	1 percent	2 percent	2 percent
+ 65.....	-	3.2	3.8
+100.....	18.8	11.7	12.7
+200.....	35.7	35.5	36.7
+325.....	16.0	18.0	17.9
-325.....	29.5	31.6	28.9
	100.0	100.0	100.0

Source of fill

	Tons
Tailing pond (hydraulic).....	28,244
Mill circuit.....	121,852
Total fill.....	150,096

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TABLE 14. - Sand-fill statistics for 280 stope 1775 level, January-December 1947 (Con.)

Comparison of sand-fill methods

	Hydraulic	Mill circuit
Total pumped.....tons	28,244	121,852
Total pumping.....hours	314.5	2,452
Average solids pumped.....percent	48.5	44.8
Average rate per 24 hours.....tons	2,155	1,193
Maximum solids pumped.....percent	51.3	47.5
Maximum rate per 24 hours.....tons	2,230	1,358
Minimum solids pumped.....percent	48.3	41.3
Minimum rate per 24 hours.....tons	2,040	1,064

Costs per ton of delivered sands, percent

	Hydraulic	Mill circuit
Rock breakage and bulkheads.....	21.4	61.3
Spill cleanup.....	.7	1.9
Drainage (pumps and lines).....	3.2	9.2
Sluicing.....	23.3	-
Pumps and pipelines.....	27.8	16.5
Desliming.....	18.0	4.1
Power.....	5.6	7.0
Total.....	100.0	100.0
Cost per ton.....cents	28.085	9.778
Average cost per ton.....do.	13.204	

TABLE 15. - Equipment specifications

Hoists:	Ore	Man
Horsepower....	600	150
Capacity.....	2,000 feet	2,000 feet.
Drum size.....	7 feet	5 feet.
Brakes.....	Dynamic	Dynamic.
Cable.....	6 x 27 wire rope, 1 1/8-inch flattened strand, independent wire rope core	6 x 19 wire rope, 1-inch flattened strand, fiber core.
Speed.....	1,200 f.p.m.	600 f.p.m.
Electric power	2,300 volts, a. c.	2,300 volts a. c.
Skips.....	85 cubic feet	-
Cage.....	-	5 x 10 feet, single deck, 20 men.
Counterbalance	-	7,000 pounds.

Slushers, production:	
Hoists.....	Double drum.
Motor.....	50 hp., 440 volts a. c.
Pull.....	6,600 pounds at 250 f. p. m., or 7,333 pounds at 225 f. p. m.
Cable.....	3 x 19 wire rope, 7/8-inch preformed.
Bucket.....	42-inch semifolding, with 3 interchangeable teeth.
Output.....	900 tons per 24 hours max. 30 tons per hour average.
Slushers, development	10-hp., 2-drum hoists; and 15-hp., 3 drum hoists.

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TABLE 15. - Equipment specifications (Con.)

Drills:

Drifters	3-1/2-inch-diameter drills on 36- and 48-inch shells.
Steel	1-inch quarter octagon.
Bits	1-1/2-inch tungsten carbide.
Stopers:	
Steel	7/8-inch hexagonal alloy.
Bits	1-3/8-inch tungsten carbide.
Longhole:	3-1/2- and 4-inch long shell leyners.
Steel	1-inch hexagonal alloy in 3- and 6-foot lengths.
Bits	2-inch tungsten carbide.

Ventilation fans:	Exhaust	Booster	Auxiliary	
Type	Axial flow	Centrifugal	Centrifugal	Centrifugal
Motor	150 hp.	50 hp.	25 hp.	15 hp.
Voltage	2,300 a. c.	2,300 a. c.	440 a. c.	440 a. c.
Speed	1,200 r. p. m.	1,200 r. p. m.	3,450 r. p. m.	3,450 r. p. m.
Capacity	105,000 c. f. m.	45,000 c. f. m.	5,000 c. f. m.	3,500 c. f. m.

Pumps, drainage:	Shaft bottom	No. 1 sump	No. 2 sump
Number of pumps	3	2	1
Motors	100 hp.	100 hp.	50 hp.
Voltage	2,300 a. c.	2,300 a. c.	440 a. c.
Speed	3,550 r. p. m.	3,550 r. p. m.	1,800 r. p. m.
Capacity, total	625 g. p. m.	600 g. p. m.	25 g. p. m.
Head, dynamic	784 feet	550 feet	500 feet

Compressors. Four compressors in parallel develop 7,000 c. f. m. at 80 to 90 p. s. i.

Power:	Volts
Main line with telephone carrier current	110,000
Mine line:	
For slushers	2,300
For trolley	440
For illumination	110

Emergency power:

Motor	900 hp., 12 cylinder, 2 cycle, supercharged marine diesel.
Generator	938 kv.-a., 2,300 volts, 235 a., 720 r. p. m., 3 phase, 60 cycle.

Sand-fill equipment:

Pumps	Station 1	Station 2	Station 3	Station 4
Number	1	2 in parallel	2 in series	2 in series
Type	Centrifugal, water seal	Centrifugal, water seal	Centrifugal water seal	Centrifugal, water seal
Size	50 hp.	20 hp.	20 hp.	20 hp.
Motor speed	1,200 r. p. m.	1,200 r. p. m.	1,200 r. p. m.	1,200 r. p. m.
Pump speed	760 r. p. m.	1,000 r. p. m.	940 r. p. m.	1,020 r. p. m.
Capacity	1,200 g. p. m.	620 g. p. m.	480 g. p. m.	480 g. p. m.
Head	60 feet	65 feet	110 feet	175 feet
Solids	35 percent	35 percent	50 percent	50 percent
Deslimers:	Primary	Secondary		
Number	2	1		
Type	Bowl	Bowl		
Size	20 by 8 ft.	20 by 8 ft.		
Speed	0.33 r. p. m.	0.36 r. p. m.		
Discharge solids	42-45 percent	50-54 percent		

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TABLE 16. - Transportation details

	1500 level	1775 level	2125 - 2500 level	Exploration headings	Development headings	Short production hauls
Locomotive:						
Type	Trolley	Trolley	Trolley	Trammer	Trammer	Trammer
Weight	13 ton	8 ton	2-4 ton in tandem	1-1/2 or 4 ton	1-1/2 ton	1-1/2 ton
Power source	Cable	Cable	Cable	Battery	Battery	Mine air
Input	250 volts d.c.	250 volts d.c.	250 volts d.c.	-	-	90-120 p.s.i.
Track:						
Rail	60 pound	40 pound	56 pound	20 pound	20 pound	20 pound
Gage	30-inch	30-inch	30-inch	24-inch	24-inch	24-inch
Cars:						
Type	Granby side dump	Granby side dump	Granby side dump	Rocker dump	Rocker dump	Rocker dump
Volume	170 cu. ft.	84 cu. ft.	84 cu. ft.	20 cu. ft.	20 cu. ft.	20 cu. ft.
Capacity	7.5 tons	3.0 tons	3.0 tons			
Cars per train	10	12	10	2-6	2-6	2-6
Average length of haul	2,965 ft.	650-1,920 ft.	2,300-3,000 ft.	Variable	Variable	50-250 ft.
Tons hauled per hour	140	190	150	Variable	Variable	12-15

TABLE 17. - Drilling data

	1938-1940	1941-1942	1943-1944	1945-1946	1947-1948	1949-1950	1951-1952	1953-1954	1955-1956
Conventional steel:									
Feet drilled	1,973,233	347,227	16,767	27,943					
Sharpenings	1,497,369	262,362	10,258	19,616					
Feet per use	1.33	1.45	1.63	1.42					
Detachable bits:									
Feet drilled	151,896	1,529,989	233,524						
New bits	(1)	180,055	38,700						
Uses per bit	(1)	4.3	3.7						
Feet per use	(1)	1.9	1.62						
Feet per bit	(1)	8.17	6.0						
C and R bits: 2/ 3/									
Feet drilled		70,354							
New bits		11,722							
Uses per bit		4.6							
Feet per use		1.3							
Feet per bit		5.98							
Throwaway bits:									
Feet drilled			890,422	395,877	1,054,668	166,016			
New bits			511,150	231,928	563,546	25,950			
Feet per bit			1.74	1.71	1.87	2.1			
New rods			(1)	3,377	8,863	803			
Uses per rod			(1)	5.05	3.69	4.2			
Feet per use			(1)	19.39	32.24	48.6			
Feet per rod			(1)	97.9	119.0	204.12			
Tungsten carbide, 1-1/2-inch bits:									
Feet drilled						875,882	796,180	778,235	611,714
New bits						5,547	4,894	3,224	2,964
Feet per bit						157.9	162.7	241.4	206.5
New rods						2,421	1,869	(1)	(1)
Uses per rod						5.72	5.33	(1)	(1)
Feet per use						63.2	79.9	93.8	120.0
Feet per rod						361.7	426.0	(1)	(1)
Tungsten carbide, 1-3/8-inch bits:									
Feet drilled							10,834	143,672	198,827
New bits							59	583	813
Feet per bit							183.6	246.4	244.6
New rods							36	(1)	(1)
Uses per rod							1	(1)	(1)
Feet per use							301	208.2	187.7
Feet per rod							301	(1)	(1)
Tungsten carbide, 2-inch bits: 4/									
Feet drilled					22,077	141,961	133,879	203,597	224,979
New bits					297	970	830	949	789
Feet per bit					74.3	146.3	161.3	214.5	285.1

1/ Not available. 2/ Incomplete record, figure estimated. 3/ Local design and manufacture.
4/ Long hole drilling.

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